NI43-101 TECHNICAL REPORT

MASBATE GOLD PROJECT

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1 SUMMARY

The Masbate Gold Project (MGP) is an operating open pit, CIL processing gold operation located in the municipality of Aroroy, Masbate Island, Region V, Republic of the Philippines. Contiguous mineral tenements held, or under application by the project total 11,593 ha.

MGP is owned and operated by three separate Philippine registered companies: Filminera Resources Corporation (FRC), Vicar Ming Corporation (VMC) and Philippines Gold Processing & Refining Corporation (PGPRC). FRC and VMC hold mineral tenements and operating permits, FRC manages open pit mining operations, while PGPRC processes the ore sold to it by FRC and VMC. CGA Mining Limited (CGA) indirectly owns 100 % of PGPRC, and 64 % of FRC and VMC.

Gold mining activity has been conducted within the MGP area since at least the 17th Century. Previous owners of the project operated an open pit / underground mining operation and associated CIL processing plant during the period 1980 to 1994 to produce over 1 M oz of gold from processing of 17 Mt of ore. The current mining operation declared commercial operation on 1st July 2009 following construction of the processing facility.

Local geology consists of a series of volcanic rocks of andesitic composition and clastic sediments, predominantly conglomerate. Gold mineralisation is hosted in a series of quartz reefs developed in a network of fault structures and deposited from hydrothermal fluids. Renewed movement along fault structure post mineralisation has deformed many of the quartz reefs and allowed the passage of low temperature fluids resulting in an overprinting of the original alteration assemblage. The MGP is classified as a low sulphidation epithermal deposit.

A series of diamond and RC drilling campaigns of mineralised deposits has been conducted at the MGP under the direction of successive owners from 1975 to the present. The most recent drilling campaign, directed by CGA, commenced in late 2009 and is ongoing. Drilling, sampling and assay data is of sufficient quality to allow estimation of mineral resources.

Data from drilling over the period 1975 to mid-2011 has been used by Mining Associates (MA) to complete a MGP mineral resource estimate as of 31st October 2011. The estimation process involved the construction of numerous vein and halo wireframes that accurately defined the shape and boundaries of mineralisation. Ordinary kriging was utilised to estimate gold grade based on drill hole assay information. The resulting block model incorporates bulk density information based on the degree of weathering and accounts for the presence of historical underground workings. Resource classification was assigned based on estimation parameters. Table 1.1 shows the resulting October 2011 MGP mineral resource estimate at a cut-off grade of 0.36 g/t Au.

Resource Category	Tonnage (Mt)	Au Grade (g/t)	Au Ounces (Moz.)
Measured	6.4	0.77	0.16
Indicated	197.8	0.76	4.84
Total Measured + Indicated	***204.3***	***0.76***	***5.00***
Inferred	102.8	0.86	2.83

Table 1.1 *October 2011 MGP combined mineral resource estimate, 0.36 g/t cut off.*

The new mineral resource estimate is an update to the May 2008 work, having incorporated new drilled results conducted since 2008 plus reconciliation with the mine production up to October

2011. The new model shows a good correlation over this period with reconciled grade control results, supporting the drop in overall grade noted both between the 2008 to 2011 estimates.

Table 1.2 shows a comparison between the new model and the 2008 model at a cut-off grade of 0.36 g/t Au. The 2011 estimate contains slightly more total resource ounces (7.83 M versus 7.77 M) and replaces the depletion due to mining since operations commenced in 2009.

Category	2011 Resource estimate			2008 Resource estimate		
	Mt	***g/t***	***Moz.***	***Mt***	***g/t***	***Moz.***
Measured	6.4	0.77	0.16			
Indicated	197.8	0.76	4.84	153.4	0.92	4.55
Total M + I	***204.3***	***0.76***	***5.00***	***153.4***	***0.92***	***4.55***
Inferred	102.8	0.86	2.83	127.2	0.79	3.22

Table 1.2 *Comparison of 2011 and 2008 MGP combined mineral resource estimates, 0.36 g/t Au cut-off.*

The 2011 mineral resource estimate has been utilised by Golder Associates to generate an MGP mineral reserve estimate as of 31st October 2011. Resources have been converted to reserves using a cost structure based on current operations, metallurgical recoveries from current processing, pit wall angles derived from independent geotechnical analysis and a gold price of USD 1,300 per oz. Reserves include a mining dilution of 5 %, with diluting material assigned a grade of 0.17 g/t, and a mining recovery of 97.5 %. Final pit designs completed by Golder consider the location of historical underground mining and geological faults.

Ore with a grade between 0.40 to 0.70 g/t Au stockpiled by FRC from commencement of mining in 2009 to 31st October 2011 is included within the reserve estimate. Table 1.3 shows the resulting October 2011 MGP mineral reserve estimate.

Resource Category	Tonnage (Mt)	Au Grade (g/t)	Au Ounces (Moz.)
Proven	5.7	0.75	0.139
Probable	105.5	0.85	2.899
Stockpiles (Probable)	7.4	0.55	0.131
Total P + P	118.6	0.83	3.168

Table 1.3 *October 2011 MGP combined mineral reserve estimate. Cut-off values; oxide material = 0.36 g/t; transitional material 0.39 g/t; primary material 0.43 g/t.*

The 2011 MGP mineral reserve estimate replaces a previous estimate completed in October 2008 (Table 1.4). The 2011 estimate contains more total reserve ounces than the previous estimate in 2008 (3.168 M versus 3.03 M) and replaces the depletion due to mining since operations commenced in 2009. Pit shells generated during both the 2011 and 2008 reserve estimates have similar footprints. The increase seen in the 2011 reserve tonnage includes an element of conversion of Inferred resources from the 2008 mineral resource estimate to higher classifications.

Category	2011 Reserve estimate			2008 Reserve estimate		
	Mt	***g/t***	***Moz.***	***Mt***	***g/t***	***Moz.***
Proven	5.7	0.75	0.139			
Probable	105.5	0.85	2.899	92.2	1.0	3.03
Stockpiles	7.4	0.55	0.131			
Total	**118.6**	**0.83**	**3.168**	**92.2**	**1.0**	**3.03**

Table 1.4 *Comparison of 2011 and 2008 MGP mineral reserve estimates.*

MGP declared commercial production on 1st July 2009. From the date of commercial production until 31st March 2012 MGP has produced over 432,000 oz. Au at an adjusted cash cost of USD 653 / oz. from processing of 13.6 Mt of ore (Table 1.5).

	2010 FY	2011 FY	2012 FY*	PTD
High Grade Ore Mined (t)	4,004,613	6,017,187	3,330,611	13,342,411
Grade (g/t)	1.36	1.14	1.05	1.18
Total Mining (BCM)	7,563,311	8,681,373	5,015,976	21,260,660
Milled (t)	4,486,408	6,152,561	3,019,627	13,658,596
Head Grade (g/t)	1.26	1.13	1.08	1.16
Recovery (%)	83.29	84.37	88.15	84.85
Gold Production (oz)	150,141	190,053	91,157	432,151
Cash Cost (USD / oz)	549	638	844	653
Cash Cost (USD / t milled)	18.35	19.86	26.24	20.71

Table 1.5 *Selected operating statistics, MGP. * = July 1 2011 to March 31 2012. PTD = Project to date. Figures in 2012 FY* reflect reduced production due to SAG Mill repair during the period July 2011 to December 2011.*

Open pit mining is conducted by Leighton Asia Limited (Leighton) under an alliance contract with FRC and PGPRC. Mining uses backhoe excavators and off highway haul trucks operating on 2.5 m bench heights. Ore is defined using dedicated RC grade control drill rigs. Current mining movements are approximately 850,000 to 900,000 BCM per month.

Processing is conducted using a conventional SABC grinding circuit and CIL. Since commissioning, mill throughput on an annualised basis has increased from 4 Mtpa to 6.5 Mtpa. The expenditure of USD 12M between September 2010 and September 2011 resulted in the ability to maintain 6.5 Mtpa throughput on blends with 65 % primary ore content. Use of lead nitrate as a reaction catalyst has maintained or improved metallurgical recovery, despite reduced residence time within the CIL circuit at the higher throughput rates. Process tailings are deposited in an engineered tailings storage facility (TSF) located 2.5 km from the Masbate Process plant.

MGP works closely with government bodies and local stakeholders to maintain all necessary permits, certificates and licences in good standing.

MGP tenements include known gold mineralisation outside of the current mineral resource and mineral reserve estimates. CGA have initiated a significant programme with annual budget of USD 20M to explore the tenement areas in addition to ongoing resource definition of near mine targets. In the opinion of the qualified persons, both exploration and continued resource definition have the potential to increase resources and reserves in future estimates.

2 INTRODUCTION

2.1 Report Preparation

This report has been prepared for CGA Mining Limited (CGA).

2.2 Purpose of the Report

This report was prepared as a Canadian National Instrument 43-101 Technical Report for CGA by qualified persons to provide updated technical information relating to the Masbate Gold Project (MGP), Philippines. The report includes an updated estimate of the mineral resource by an independent qualified person (Andrew Vigar, Mining Associates) and an updated estimate of the mineral reserve by an independent qualified person (Stephen Jones, Golder Associates).

2.3 Information Used

The quality of information, conclusions and estimates contained within this report is based upon:

(i) information available internally at the time of the report;
(ii) data obtained from outside sources, referenced at the end of the report; and
(iii) the assumptions, conditions and qualifications set forth in the report.

In the opinion of Mining Associates (MA), CGA provided open access to all records necessary to enable proper assessment of the project and mineral resource estimate. MA also relied on information provided in previous NI 43-101 technical reports on the MGP detailed below.

"Technical Report on the Mineral Resources of the Masbate Deposit, Masbate Province, Republic of the Philippines held by Thistle Mining Inc. through Filminera Resources Corporation", dated 28th February 2006 and authored by Mining Associates.

"Technical Report on the Mineral Resources of the Masbate Deposit, Masbate Province, Republic of the Philippines for CGA Mining Limited", dated 20th May 2008 and authored by Mining Associates.

"NI43-101 Technical Report, October 2008 Masbate Gold Project, Masbate Island, Philippines", dated 5th December 2008 and authored by Lower Quartile Solutions Pty Ltd.

In the opinion of Golder Associates (Golder), CGA provided open access to all records necessary to enable proper assessment of the project and mineral reserve estimate.

2.4 Qualified Persons

The Technical Report is prepared by Mark Turner (CGA), Andrew Vigar (MA) and Stephen Jones (Golder). Details on individual sections of responsibility within the report and site visits conducted in relation to the construction of the technical report are presented in Table 2.1.

Qualified Person	Site Visit	Sections of Responsibility
Mark Turner	Regular visits	1, 2, 3, 4, 5, 6, 7, 8, 9, 10, 11, 13, 16, 17, 18, 19, 20, 21, 22, 23, 24, 25, 26, 27
Andrew Vigar	August 2011	1, 10, 11, 12, 14, 25, 26
Stephen Jones	November 2011	1, 15, 25, 26

Table 2.1 *Qualified Person sections of responsibility within the report and details of site visits.*

3 RELIANCE ON OTHER EXPERTS

The authors of this Technical Report have not relied upon other experts in its preparation, other than obtaining input from persons employed within CGA who have provided information regarding legal, environmental or other matters relevant to this report.

4 PROPERTY DESCRIPTION & LOCATION

4.1 Area of Property

MGP consists of contiguous tenements totaling some 116 km^2 that extend approximately 16 km in a northwest – southeast mineralized corridor.

4.2 Location

MGP is located within the Republic of the Philippines near the northern tip of the island of Masbate, 360 km southeast of the country's capital of Manila. MGP is centred at latitude 12^O 28' N and longitude 123^O 24' E (Figure 4.1). The project area lies within the municipality of Aroroy, Masbate Province, Region V.

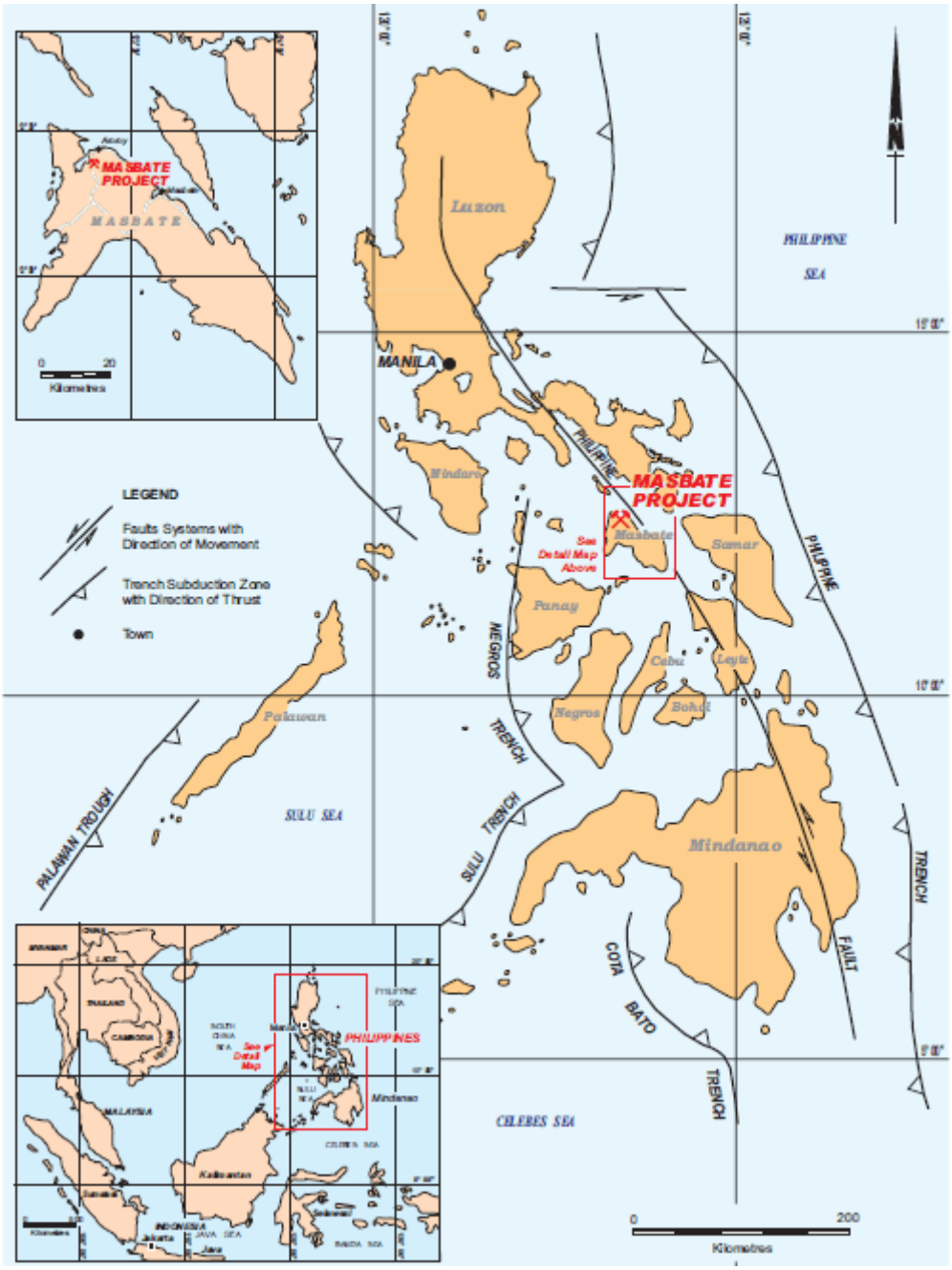


Figure 4.1 *Location of the MGP. (BDA 1998)*

4.3 Tenure

MGP holds a contiguous set of mining or exploration claims and applications that together cover an area of approximately 11,593 ha (116 km^2). The tenements jointly cover all principal mining areas and are illustrated in Figure 4.2. Details of individual tenements are provided in Table 4.1 to 4.5.

Description	No. of Claims	Area (ha)	Term	Expiry Date	Remarks
Patented Mineral Claims	29	236.03	Perpetual	None	Contiguous
Mineral Lease Claims, MRD 406	12	108.00	25 Years	17/11/2013*	
Total	41	344.03			

Table 4.1 Patented Claims and Mineral Claims, MGP. * Will be converted to an MPSA on expiry as per the 1995 Mining Act.

MPSA Identification	Holder	Area (ha)	Date Approved	Date Expiry	Remarks
MPSA-095-97-V	FRC	285.68	20/11/1997	19/11/2022	Commercial Operation
MPSA-255-2007-V	FRC	129.72	30/07/2007	29/07/2032	Commercial Operation; Approved DMPF
MPSA-256-2007-V	FRC	126.28	30/07/2007	29/07/2032	Exploration
MPSA-329-2010-V	FRC	584.20	23/03/2010	22/03/2035	Exploration
MPSA-219-2005-V	VMC	785.57	20/10/2005	19/10/2030	Exploration
Total		1,911.45			

Table 4.2 *Granted Mineral Production Sharing Agreements (MPSA), MGP. FRC = Filminera Resources Corporation; VMC = Vicar Mining Corporation.*

Tenement ID	Holder	Date Approved	Date Expiry	Area	Remarks
EPA-010-2010-V	FRC	05/04/2010	04/04/2016	5,231.39	Ongoing exploration

Table 4.3 *Granted Exploration Permits, MGP*

Tenement ID	Holder	Date filed	Area	Commodity	Remarks
APSA V-0030	FRC	16/07/1991	1,210.11	Gold, Silver etc.	Endorsed to MGB-CO. Under final evaluation. Published May 1997
APSA V-0032	FRC	16/07/1991	86.39		
APSA-000277-V	FRC	23/08/1999	63.00		Additional doc filings

Table 4.4 *Mineral Production Sharing Agreements (MPSA) under application, MGP.*

Tenement ID	Holder	Date filed	Area	Remarks
EXPA-000167-V	FRC	09/23/2010	1,937.26	Additional doc filings
EXPA-000100-V	FRC	09/23/2010	809.43	Additional doc filings

Table 4.5 *Exploration Permits under application, MGP.*

Tenure systems in the Philippines are defined under the 1995 Philippines Mining Act and administered by the Department of the Environment & Natural Resources (DENR).

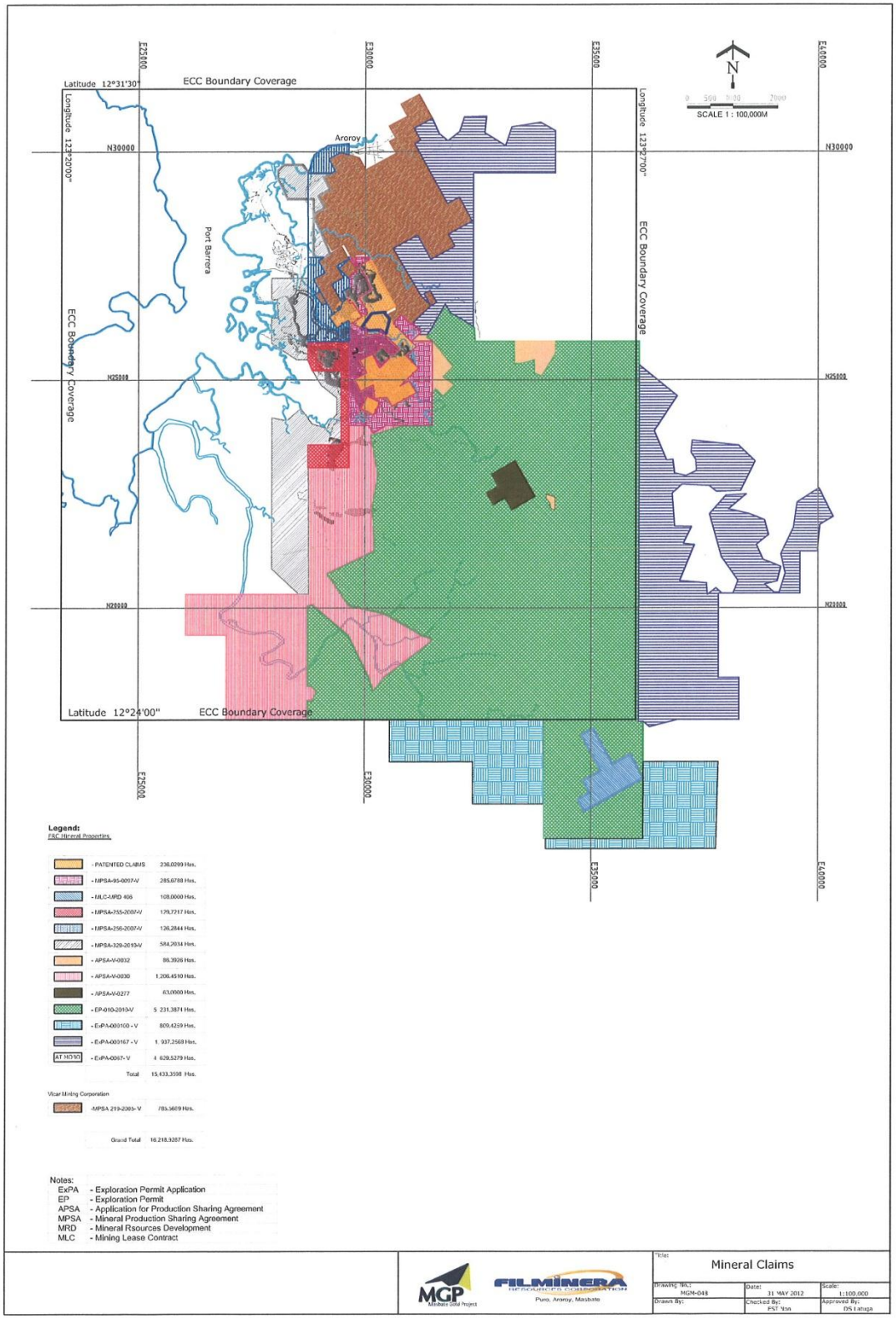


Figure 4.2 *Mineral Claims, MGP*

4.4 Masbate Gold Project – CGA Group ownership

The MGP is owned by three Philippine registered companies, Vicar Mining Corporation (VMC), Filminera Resources Corporation (FRC) and Philippine Gold Processing and Refining Corporation (PGPRC).

FRC owns the majority of the Masbate Gold project tenements and is responsible for the mining, environmental, social and community relations on the project site. VMC owns MPSA 219-2005-V which includes Pajo deposit. FRC manage and operate this property under a Technical Services Agreement between FRC and VMC.

PGPRC holds a minerals processing permit, has developed and owns the process plant and is responsible for the sale of all gold. PGPRC, FRC and VMC have a contractual relationship, which includes PGPRC purchasing ore from FRC and VMC whilst maintaining joint financial and legal liability for social and environmental obligations under Philippine law. To date, all ore processed by PGPRC has been purchased from FRC owned MPSA, mineral lease and patented mineral claims.

CGA Mining Ltd (CGX:ASX; CGA:TSX) indirectly owns 100% of PGPRC and indirectly owns 40% of FRC with the remaining 60% being owned by Zoom Minerals Holdings Inc., in which CGA has a 40% interest (i.e. CGA has 64% indirect interest in FRC). CGA also indirectly holds 40% of VMC with the remaining 60% being owned by Aroroy Resources Inc., in which CGA has a 40% interest (i.e. CGA has a 64% indirect interest in VMC). The CGA group ownership structure for the Masbate Gold Project assets is illustrated in Figure 4.3.

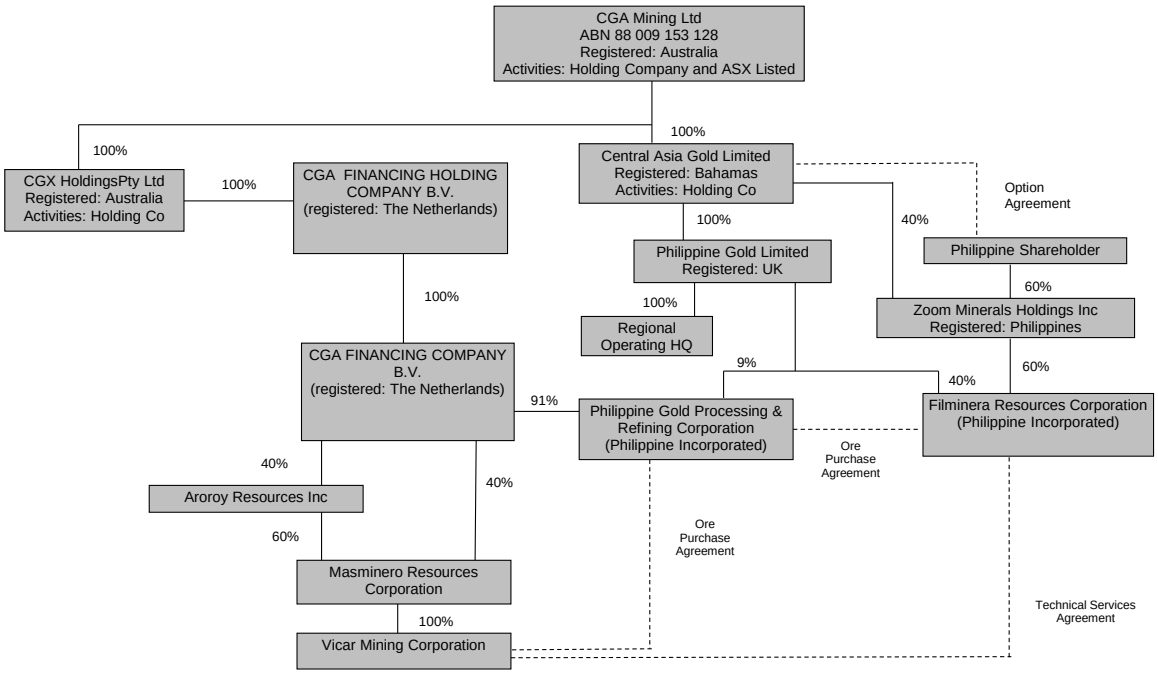


Figure 4.3 *CGA group ownership, MGP (CGA 2012)*

4.5 Land Ownership and Access

FRC holds the surface rights to all current mining pits, waste dumps and stockpiles, the Masbate Process plant, Tailing Storage Facility (TSF) and other vital infrastructure facilities (causeway port, airstrip, housing compound, tailings lines, etc. – Refer to Figure 4.4).

4.6 Royalties & Agreements

There is no mineral royalty, but a 2 % excise on gross gold and silver sales is payable to the Philippine government under the MPSA model.

There is no Indigenous People (IP) associated with the MGP.

Legislation review has recently amended the compulsory minimum SDMP contribution from 1 % of direct mining and milling costs to 1.5% of total operating costs. The Masbate Gold Project has an existing agreement with the DENR whereby mandated SDMP commitment is capped at 1% of direct milling and mining costs for a 5 year period following the commencement of commercial production (1st July 2009).

4.7 Environmental Liabilities

Base line environmental surveys conducted before the commencement of operations as part of the Masbate EIA are accepted by the Department of Environment and Natural Resources (DENR) as the basis for future environmental assessment. FRC and PGPRC contribute to a Mine Rehabilitation and Decommission Fund administered by a board comprising members of the FRC, PGPRC, DENR and local elected officials. Final land use planning has been submitted to the DENR and approved.

Part of the requirements of an EP is the submission of an environmental work programme (EWP) with a budget equal to at least 10% of the total budget of exploration activities.

4.8 Permits

The MGP holds all the permits, certificates, licences and agreements required to operate the project. There are currently 298 separate operating permits (not including importation permits for individual reagents to the Philippines). The most strategically significant permits include (but are not limited to):

Mineral Production Sharing Agreements (MPSA): MPSA 95-97-V, MPSA 329-2010-V, MPSA 256-2007-V, MPSA 255-2007-V, MPSA 219-2005-V
Exploration Permit (EP): EP 010-2010-V
Environmental Compliance Certificate (ECC): ECC No. 9804-003-120c
Mineral Processing Permit (MPP): MPP No. 10-2007-V
Waste Water Discharge Permit: WDP-11L-05MA-029
Power Station – permit to operate: TPOA-12D-05MA-325
Causeway Port facility – Registration: PTO No. 175
Airstrip – Registration: AGA-R-001-2011
Special Land Use Permits: SLUP No. R5MBT-031, SLUP No. R5MBT-032, SLUP No. R5MBT-033

The Philippines has an established framework that is well regulated and monitored by the DENR and other regulatory bodies. FRC, PGPRC and VMC have a dedicated programme that monitors permit compliance and work closely with authorities to promptly address additional requests for information. Permits, certificates and licences are issued for various periods and need to be regularly reviewed.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE & PHYSIOGRAPHY

5.1 Topography & Elevation

The Masbate Gold Project is situated in a series of hills (maximum elevation 705 mRL) with moderate to steep slopes, often deeply incised by rivers and streams. A narrow coastal plain is present in parts of the property located immediately adjacent to the bay. Figure 5.1 illustrates the general topography of the project.

Figure 5.1 *Typical topography and vegetation of the Masbate Gold Project. A series of deeply incised hills with moderate to steep slopes rises abruptly from a narrow coastal plain. The airstrip associated with the Masbate Gold Project is visible in the central foreground, with Amoroy Barangay located on the narrow beach in the central right foreground. Mining activity on the central hill in middle distance is the Colorado deposit. Coastal mangrove swamps progress to modified forest, secondary regrowth and plantations on the coastal plain and hill slopes. Photograph taken looking to the southeast. (CGA 2012)*

5.2 Vegetation

Vegetation is dominated by modified forest or secondary regrowth. Plantations (predominantly coconut and banana) are present on many of the lower in land hill slopes. Areas of grassland are present on the coastal plain and lower slopes of hills close to the coast, especially in southern sections of the property. Mangrove swamps are present on some coastal sections (Figure 5.1).

5.3 Access to the Property

MGP operates a converted fishing trawler (Lady Jacqueline – Figure 5.2) to transport personnel and small freight items to / from a port facility located close to the mine site.

Figure 5.2 *Lady Jacqueline, a converted fishing trawler associated with the MGP and one of the primary transport methods used to access the property. (CGA 2012)*

The most common route for personnel travelling to MGP from Manila involves a commercial flight to Masbate City, followed by boat travel directly to site along the north-east coastline of Masbate Island (Figure 5.3).

A common alternative involves a commercial flight to the Legazspi City, 40 km road transport to the port of Pilar and boat travel to site across the stretch of Philippine Sea separating the islands of Luzon and Masbate.

Sealed roads link Manila to Legazpi City (500 km via the Pan-Philippine Highway AH26) and Legazpi City to Pilar (40 km via combined Pan-Philippine Highway AH26 and the Putiao – Pilar – Donsol – Sorsogon Road).

A predominantly sealed provincial road (Central Nautical Highway) links the local population centre of Aroroy to Masbate City (60 km; Figure 5.3).

Masbate City is linked to other ports in the Philippines via commercial passenger and freight ferry. Regular ferries service Masbate City from Pilar – Sorsogon (daily) and Cebu (3 times per week). Two commercial airlines operate daily flights to / from Masbate City.

A Civil Aviation Authority of the Philippines (CAAP) registered air strip and helipad are located close to the Masbate Gold Project (Figures 5.1, 5.3). Charter flights to Manila take approximately 1 hour and the strip can accommodate light aircraft holding up to 19 passengers.

Large freight items, bulk fuel and reagents are commonly transported to the Masbate Gold Project port facility via commercial barges chartered from commercial ports in the Philippines. Barge travel time between the port of Manila and site is approximately 40 hours one way.

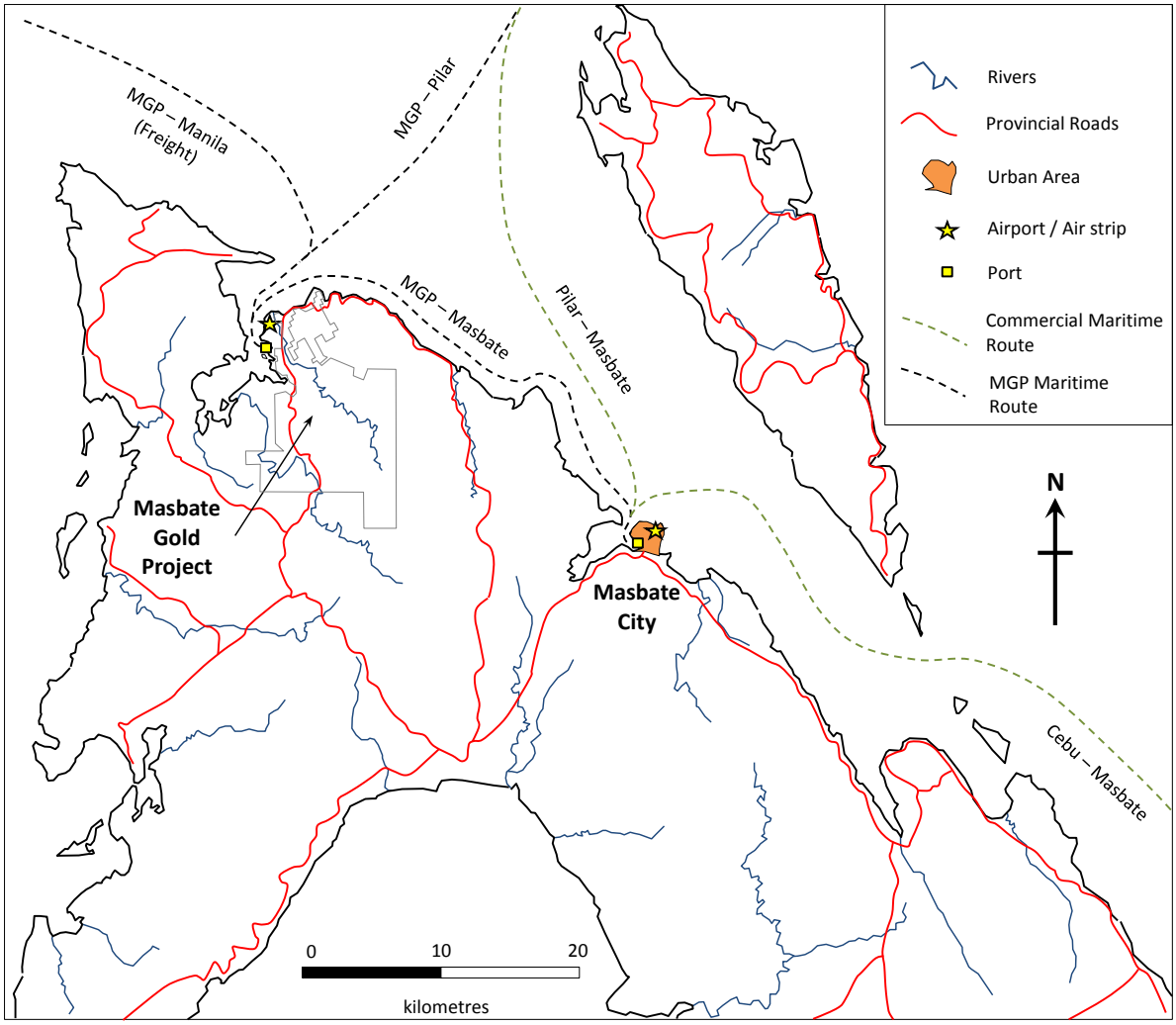


Figure 5.3 *Access routes to MGP. (CGA 2012)*

5.4 Proximity to Population Centres

The province of Masbate consists of three major islands; Masbate, Ticao and Burias; with a total population of approximately 800,000 people. Masbate Province is subdivided into twenty municipalities and one city (Masbate City). The City of Masbate is the provincial capital with a population of over 80,000 people in 13,400 households (2007 Census figures).

MGP is located within the municipality of Aroroy. Aroroy is the most populous municipality of Masbate Province, with a total population 72,393 and a local economy dominated by agriculture, fishing and mining.

The major population centre within the municipality of Aroroy is the township of Aroroy which is located 5 km from the MGP. Politically, the municipality of Aurory is subdivided into forty one barangays.

A barangay is the smallest administrative division in the Philippines and is the native Filipino term for a village, district or ward. The MGP straddles the boundaries of several barangays within the municipality of Aroroy, eight of which have been deemed to significantly host activities of the mining

project and are eligible for SDMP funding (Section 20.5). Total population of the eight impact barangays is approximately 24,000 people.

5.5 Climate and Operating Season

The climate of the Philippines is tropical. Rain may fall throughout the year but the climate typically has a distinct period of higher rainfall (wet season). The wet season commences during June and lasts until November when the entire Philippines is under the influence of winds associated with the southwest Monsoon. During the period December to May, winds switch to the north east trades which are associated with significantly reduced rainfall. High average daily temperatures (30 to 35°C) and high atmospheric humidity are present throughout the entire year.

The climate of Masbate is characterised by a short 'dry' season lasting from one to three months during the period February to April and more frequent rainfall during the remainder of the year (Type III under the modified Corona classification, Figure 5.4). Comparison between annual rainfall records can be highly variable and influenced by unseasonal tropical depressions. Rainfall associated with individual tropical depressions or typhoons may exceed 100 mm.

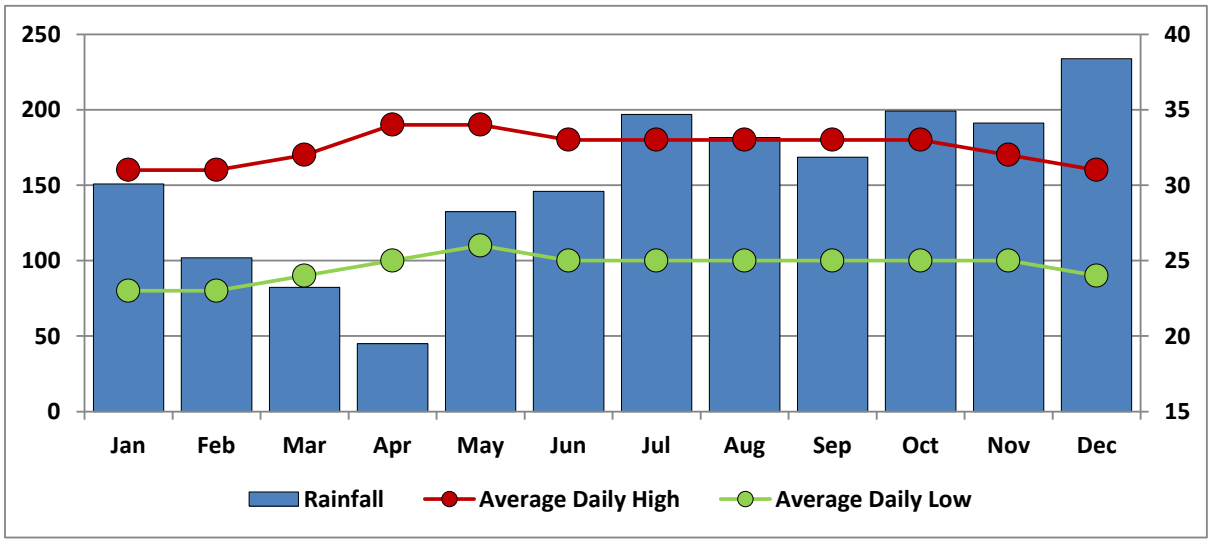


Figure 5.4 *Average monthly rainfall and temperature data for Masbate. Data sourced from worldweatheronline.com (CGA 2012)*

Mining activities at the MGP are not significantly affected by the climate. Heavy rainfall may temporarily affect access to areas of active exploration.

5.6 Typhoons

The Philippines is affected by typhoons on an annual basis. Typhoons are most common during the period from June to November but occasionally occur at other times of the year. Typhoons commonly affect the northern island of Luzon and are less common in the islands in the southern section of the Philippine archipelago.

29 individual typhoons have passed within a 50 km radius of the MGP since 1950, an average of approximately 5 per decade (Figure 5.5). The most recent typhoon to pass within 50 km of the MGP occurred during the late stages of construction and commissioning in June 2009. Although the surrounding area was affected by minor flooding and wind damage to trees, operations at the MGP

were largely unaffected. The MGP monitors typhoon and tropical storm development and progress, and has developed emergency planning to protect personnel and equipment in the event of a typhoon passing close by.

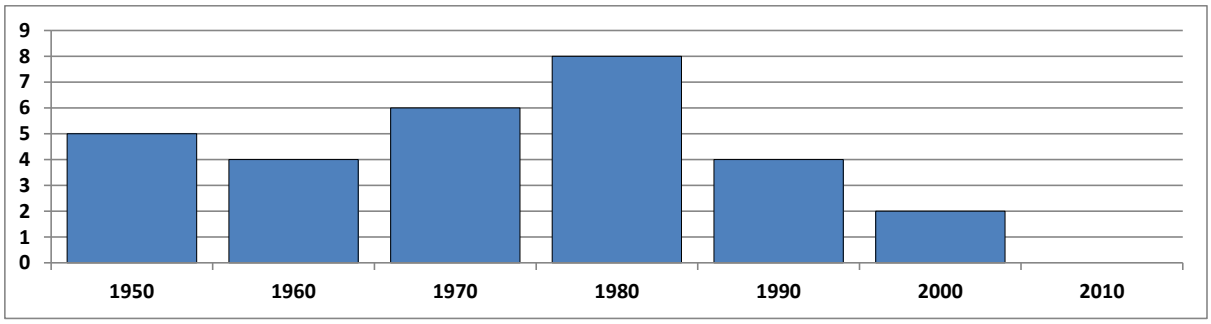


Figure 5.5 *Frequency (by decade) of typhoons passing within 50 km of the MGP, 1950 to 2012. Data sourced from Digital Typhoon website. (CGA 2012)*

5.7 Infrastructure

5.7.1 Sufficiency of Surface Rights

MGP holds sufficient surface rights to conduct current mining operations. Active negotiation with individual landowners holding remaining small parcels within the footprint of future dumps is ongoing.

5.7.2 Power

MGP includes a 30 MW power station located immediately adjacent to the Masbate Process Plant. The power station consists of a number of generating sets and provides a reliable power source sufficient to meet current operations.

The power station design allows the addition of more generating sets or the replacement of individual sets to units of higher generating capacity in the event of the expansion of MGP operations requiring additional power. MGP is currently licenced to operate a power station up to 50 MW generating capacity.

5.7.3 Water

Water is currently drawn from the Guinobatan River and is sufficient to supply current operations. The MGP holds permits to draw additional water from both the Lanang and Bangon Rivers if additional water is required. The MGP also has the ability to draw water from the Fresh Water Reservoir located immediately adjacent to the current TSF.

5.7.4 Mining Personnel

The MGP sources the majority of its employees from the province of Masbate. FRC and PGPRC have committed to maintain at least 70 % local employees and require long term major contractors to the MGP to do the same. A long history of small scale gold mining activity ensures a ready supply of suitable semi-skilled workers is available in local communities.

Positions requiring skills and experience not available locally are filled from throughout the remainder of the Philippines. There are a small number of highly skilled and experienced expatriate employees present at the MGP. Expatriates actively mentor and assist in the skill development of Filipino employees.

5.7.5 Communications

A digital satellite communications package provides telephone, email and facsimile coverage to the MGP. Mobile telephone coverage is available throughout the majority of the mining and exploration areas.

A multi-channel radio network is utilised for operations communication within the mine and process plant.

5.7.6 Mining Infrastructure

The MGP has a well-developed mining infrastructure sufficient to support current operations. Further details of mining infrastructure are detailed in Section 18.

6 HISTORY

6.1 Mining History

Small scale gold mining has been carried out in Masbate for several centuries. Mining for gold took place under Spanish colonisers and early Spanish settlers recorded previous activities of local and Chinese miners in the district.

Small scale mining continued sporadically in succeeding centuries. Mining interest in the area now covered by the MGP was revived by the arrival of American prospectors in the early years of the 20th Century. As many as six separate mining operations were active in the area through the 1930's with stamp batteries and cyanide mills treating 50 to 400 tonnes per day. Mining was primarily underground operations with selective mining of narrow, high grade quartz veins.

Commercial mining ceased following the start of the Second World War. Significant mining activity resumed in 1979 when Masbate Gold Operations (MGO), a subsidiary of Atlas Consolidated Mining and Development Corporation ("Atlas"), constructed a mill and commenced open pit and later underground mining. Commercial gold production started in April 1979. Over 1 million oz. Au were produced over 14 years of operation. Atlas operations ceased in 1994.

6.2 Prior Ownership

Prior to 1935, a succession of small American and Filipino mining companies owned and operated a number of mines in the area of the MGP. Over time, the number of operating companies within the current property diminished as consolidation of deposits took place.

Further consolidation took place on March 9th 1935, when Atlas Consolidated Mining & Development Corporation was incorporated as Masbate Consolidated and Mining Company, Inc. as a result of the merger of assets and equities of three companies, namely, Masbate Consolidated Mining Company (Masbate), Antamok Goldfields Mining Company (Baguio) and IXL Mining Company (Masbate). Masbate Consolidated and Mining ceased operations at Masbate in 1941 due to the Second World War.

In 1979, Atlas formed Masbate Gold Operations (MGO), constructed a mill and began mining operations.

In 1995, London Fiduciary Trust PLC, later renamed Philippines Gold PLC (PGO) agreed with Atlas to purchase 100% of MGO, the mines, claims and facilities free of debts and obligations for USD 10M. MGO claims and assets were transferred to Base Metals, later renamed Filminera Resources Corporation (FRC).

During 1999 and 2000, a series of corporate transactions saw the eventual acquisition of PGO by Thistle Mining.

On January 31, 2007 the CGA entered into a Sales and Purchase Agreement (SPA) for the acquisition of 100% of Thistle's interest in the MGP. The agreed purchase consideration was USD 51 million and the transaction was completed on March 19, 2007, with CGA paying cash totalling USD 24.5M (USD 0.5M had already been paid as a deposit) and issuing 40,985,538 shares to a wholly owned

subsidiary of Thistle, Toowong Mining BV. As a result of the transaction, CGA indirectly owned through its 100% ownership of PGO, 40% of FRC and 100% of PGPRC.

6.3 Previous Work

Atlas conducted a limited drilling campaign (< 3,000 m) at Main Vein, Colorado and Binstar as part of a feasibility assessment before commencing operations.

During 1980 to 1994 Atlas completed over 50,000 m of drilling within the Masbate resource area, much of it during the period 1989 to 1991. Atlas also conducted some geological mapping and sampling of creeks and streams within the area of the current MGP.

Base Metals (funded by PGO) undertook an intensive drilling programme during 1997 (total of 30,000 m) to enable resource definition for their 1997 feasibility study. A small follow up programme was drilled in 1998 with the aim to upgrade resources from Inferred category. In all, Base Metal drilling focussed on 6 prospects – Main Vein, Libra, Holy Moses - Basalt, Colorado, Grand View and Montana – these areas were primarily where Atlas records had indicated significant potential.

During the period 2002 to 2005, FRC (funded by Thistle Mining) completed over 26,000 m drilling in four separate campaigns relating to successive resource estimate, a scoping study and feasibility study. Extensive surface mapping was also completed within the project area at this time.

During 2007 and 2008 FRC (funded by CGA) drilled a further 22,000 m, some of which related to sterilisation of the process plant site and TSF.

6.4 Historical Resource Estimates

A number of resource estimates have been completed for the current area of the MGP (Table 6.1). Direct comparison between individual estimates is difficult as they have been complied using different estimation techniques, include different combinations of individual deposits and assume different mining methods. Estimates completed before 2004 are not considered to be NI43-101 compliant. The qualified persons have not done sufficient work to classify historical estimates (pre 2004) as current mining resources and CGA is not treating historical estimates (pre 2004) as current mineral resources.

Date	Company	Cut Off	Measured + Indicated			Inferred			Total		
		g/t Au	Mt	g/t	Moz.	Mt	g/t	Moz.	Mt	g/t	Moz.
1994*	Atlas	1.5 O/P:2.0 U/G	7.1	2.51	0.57	4.0	2.86	0.37	11.2	2.65	0.95
1996*	ACA Howe	1.5 O/P:2.0 U/G	7.8	2.53	0.63	5.5	2.25	0.40	13.8	2.42	1.03
1997*	Base Metals	0.70							33.7	1.46	1.77
1998*	Base Metals	0.70	35.5	1.53	1.75	13.9	1.42	0.63	49.3	1.50	2.38
1998*	Base Metals	0.70	53.1	1.37	2.34	18.2	1.27	0.74	71.3	1.34	3.08
2002*	FRC	0.70	49.0	1.41	2.22	16.2	1.33	0.69	65.2	1.39	2.91
2004	IMC	0.70	44.5	1.34	1.92	20.0	1.43	0.92	64.5	1.34	2.84
2006	MA	0.50	94.6	1.19	3.62	59.8	1.18	2.27	154.4	1.19	5.89
2008	MA	0.36	153.4	0.92	4.55	127.2	0.79	3.22	280.6	0.86	7.77

Table 6.1 *Previous estimates of Mineral Resource for MGP. * Historic estimates not considered to be NI43-101 or JORC compliant. FRC = Filminera Resources Corporation; IMC = Independent Mining Consultants Pty Limited; MA = Mining Associates Pty Limited. Mineral resources that are not mineral reserves do not have demonstrated economic viability.*

Resource estimates by Atlas and ACA Howe concentrated on selective mining approaches not comparable to the current scale of mining. Underground resource and reserve estimates were restricted to limited areas defined largely by channel sampling from mine workings with drilling mainly used only used as a guide for development. Both estimates used an underground cut-off grade of 2.0 g/t and a top cut value of 10.0 g/t Au. Open pit resources used a cut-off grade of 1.5 g/t, a top cut of 5 g/t Au and polygon estimation technique. Estimation utilized data from surface mapping and trenching, diamond drilling, RC drilling, blast hole drilling, and channel sampling along drifts, cross cuts and raises.

Base Metals undertook a Feasibility study in 1997 that included only the central resource areas of Main Vein, Holy Moses – Basalt, Colorado and Montana. The 1997 estimate relied on polygonal interpretation and blocking based on section analysis and grade estimation using a nearest neighbor technique. The resources quoted in the Feasibility study did not include a breakdown of classification. During the study, it became apparent that a larger, low grade open cut operation was more feasible and that restarting the existing underground mining operations was not warranted.

With more drilling, Base Metals updated the feasibility study in 1998, still using the nearest neighbor estimation technique. This approach was criticized by independent reviewers and a second estimate in 1998 included block kriging methods and addressed some concerns with 1997 assay data.

After additional drilling completed in 2002, FRC updated the resource estimate for Colorado, Grand View and Montana (Main Vein remained unchanged). The estimates were similar to Base Metals and used 3D wireframe interpretations of individual veins plus block kriging.

The 2004 Independent Mining Consultants Pty Limited (IMC) estimate was undertaken using multiple indicator kriging over broad domains and used all drilling information up to the end of 2003. The estimate did not include resources for the Binstar deposit.

Mining Associates Pty Limited (MA) completed a resource estimate in 2006 following a detailed review of drill hole, assay and geological data from drilling to the end of 2005. MA recognized that individual veins were often surrounded by a zone of lower grade mineralisation they termed “stockwork” and that both vein and stockwork material may be economic to process under a bulk open pit mining scenario. Wireframes for both individual veins and areas of stockwork were created

and grades estimated using the ordinary kriging technique. The 2006 estimate included resources for the Binstar deposit.

MA further updated the resource estimate in 2008 following additional drilling completed in 2007 and early 2008. For the 2008 estimate MA added an additional large domain, termed "Halo", which encompassed the entire mineralized halo for all areas at a nominal cut-off of 0.1 g/t. Grades within the Halo were estimated using ordinary kriging (as were individual vein and stockwork domains).

The 2008 estimate for the first time also included resources for several large dumps of spoil material from previous mining operations (pre Second World War and Atlas) that exist on the MGP tenement, as well as some areas of mineralization present in slope deposits derived from the shedding of material from out cropping veins (eluvial deposits). Together dump and eluvial resources total 18.5 Mt @ 0.62 g/t Au of Indicated category material at a cut-off grade of 0.36 g/t.

6.5 Historical Reserve Estimates

Lower Quartile Solutions Pty Limited (LQS) utilised the MA 2008 resource estimate to complete a reserve estimate in October 2008 (Table 6.2).

Date	Company	Cut Off	Proven			Probable			Total		
		g/t Au	Mt	g/t	Moz.	Mt	g/t	Moz.	Mt	g/t	Moz.
2008	LQS	0.40				92.2	1.0	3.03	92.2	1.0	3.03

Table 6.2 *Previous estimates of Mineral Reserve for MGP. LQS = Lower Quartile Solutions Pty Limited.*

The reserve was calculated using a USD 750 per ounce, included mining dilution of 5% and mining recovery of 95%. Predicted mining, processing and administration costs, plus predicted metallurgical recovery was used to calculate an economic cut-off grade of 0.4 g/t Au.

6.6 Previous Production

Pre-Second World War production figures are not available. Atlas production over the period 1980 to 1994 totalled 1.03 Moz. Au and 0.99 Moz. Ag from mill processing of 17.4 Mt of ore (Table 6.3). A small amount of additional gold production (0.05 Moz Au) was recovered from Heap leach processing of an unknown amount of low grade ore

Atlas open pit operations were commissioned in 1979, with underground operations commencing in 1986. Mill throughput averaged 3,500 tonnes per day and utilised CIP gold recovery. Small heap leach trial operations were conducted between 1984 and 1993.

Year	Ore Milled		Au Oz. Produced			Ag Oz. Produced
	Tonnes (t)	Grade (g/t)	Mill	Heap Leach	Total	
1980	990,542	2.13	50,030		50,030	36,566
1981	1,211,409	2.39	81,475		81,475	72,254
1982	1,259,700	2.28	80,201		80,201	65,071
1983	1,271,560	2.31	82,824		82,824	61,137
1984	1,266,591	2.26	79,881	1,470	81,351	71,931
1985	1,345,609	2.06	78,595	6,301	84,896	56,725
1986	1,255,817	2.08	72,987	4,974	77,961	75,487
1987	1,224,929	1.76	58,091	5,235	63,326	80,972
1988	1,265,913	1.88	65,175	5,392	70,567	91,344
1989	1,319,528	2.09	77,549	5,825	83,374	91,084
1990	1,311,084	2.16	79,977	7,031	87,008	76,602
1991	1,286,002	2.10	76,760	6,539	83,299	61,221
1992	1,224,288	2.07	74,803	3,043	77,846	53,411
1993	946,084	2.09	56,374	3,572	59,946	40,555
1994	217,768	2.29	14,222		14,222	10,114
Total	**17,396,824**	**2.12**	**1,028,944**	**49,382**	**1,078,326**	**944,474**

Table 6.3 *Atlas production data from processing operations, MGP, over the period 1980 to 1994.*

Ore recovered from open pit mining of Main Vein produced the vast majority of ore for processing. Over the 14 years of operations, Atlas mined material from a total of 21 open pits, 4 alluvial and 4 underground prospects, although the majority of these supplied minor quantities of ore. At the time of closure, ore was being actively mined from Main Vein, Montana, Holy Moses, Colorado Vein 5 and Colorado Vein 9.

7 GEOLOGIC SETTING & MINERALISATION

7.1 General

Description of the geology of the MGP is summarised below from a report by Jigsaw Geoscience Pty Limited (Jigsaw). Jigsaw completed a comprehensive geological campaign of the MGP in 2010 which included diamond core logging, field mapping, structural analysis, multi-element geochemistry and spectral analysis of clay and mica minerals.

7.2 Regional Geology

Masbate Island lies in the central part of the Philippine archipelago, midway between the islands of Mindanao and Luzon. The Philippine archipelago has evolved via subduction, plate collisions and strike slip faulting which commenced during the Mesozoic and is ongoing. They consist of accreted volcanic arcs, marginal basins, ophiolites and continental fragments, formed in response to convergence between the South China Craton and the Pacific Plate.

At present there are 3 active elements in the Philippine archipelago (Figure 4.1). In the west, a gentle east-dipping trench system comprising the Manila Trench in the north and the Negros and Cotabato Trenches in the south. The east side of the Philippines is bordered by west-dipping subduction involving the Luzon Trench and Philippine Trench. The third active element is the Philippines Fault, a sinistral strike-slip fault system which lies midway between the 2 subduction systems and roughly parallels them in strike. The Philippine Fault lies just offshore of the MGP and is inferred to come ashore along the eastern coast of Masbate Island not far from Masbate City. The Philippine Fault is believed to have been active since the Late Pliocene (4 – 2 Ma), reactivating pre-existing structures, and is still highly seismically active with 10 large earthquakes in the past 100 years.

7.3 Local Geology

The local stratigraphic sequence has been described from several geological investigations and is illustrated schematically in Figure 7.1. Stratigraphic relationships however, are not consistent across the region and no author has provided any definitive age dating or sources to support the ages of rock units described here.

In the northwest region of Masbate, the oldest recorded rocks are inferred to be pillowed mafic lavas of the Mt Manapao Basalt, which lie east of the MGP near the east coast of Masbate Island. These are capped by deep-water mudstone of the Boracay Formation and are estimated to be Cretaceous or older. Unconformably and / or structurally juxtaposed against these rocks are sandstones, ‘red’ calcarenites and volcanics of the Late Eocene to Oligocene Mandaon Formation. Both the Mt Manapao and Mandaon Formation are intruded by the Aroroy Diorite, a large intrusive complex of inferred Middle Oligocene age. The Mandaon Formation sediments are unconformably overlain by the shallow marine clastics, andesitic volcanics and carbonate platform rocks of the Sambulawan Formation, inferred to be of Late Oligocene – Early Miocene age. The Late Miocene to Early Pliocene Masbate Group overlies the Sambulawan Formation and comprises various andesitic volcaniclastics subdivided into the basal Lamon Andesite, Buyang Limestone, Lanang Conglomerate and Cawayan Clastics. Mt Nabongsoran Andesite intrudes various clastic lithologies and is included

within the Masbate Group. The Late Pliocene – Early Pleistocene Bugui Limestone caps the sequence.

Two NNW-striking fault structures are identified within the MGP: the Pinanaan Fault in the east coincides with the Pajo Reef, and the Malubi Fault in the west which has the David Sun Veins developed within it. These faults are inferred to be a 'sinistral wrench couple' between which the Masbate Reef system ruptured, and are inferred to be sympathetic shears to the Philippine Fault.

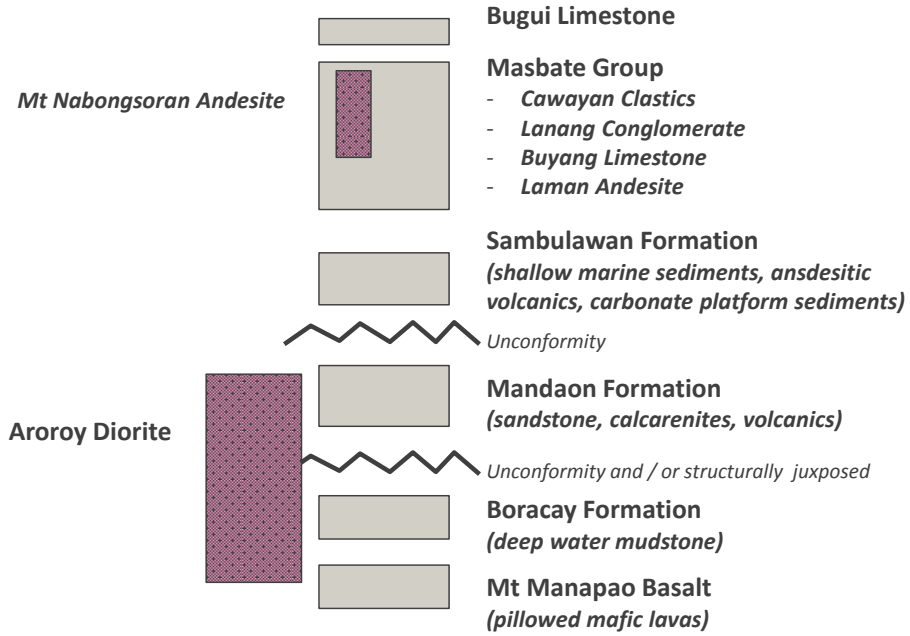


Figure 7.1 *Schematic stratigraphy for northwest Masbate Island. Illustrated from descriptions documented by Standing and Halley (2010) and Vigar (2011). (CGA 2012)*

7.4 Property Geology

7.4.1 Lithology

The main rock types present at the MGP include andesite, basaltic andesite and rocks derived almost exclusively from andesite volcanics; ploymict clastic rocks derived from multiple source regions; and porphyritic intrusives of andesite composition.

Andesitic Rocks

Massive, coherent feldspar-phyric andesite, often intermingled with andesite breccia, is common in diamond drill core and outcrop. In several instances, the andesite is pillowed and contains abundant amygdales of carbonate, and has inert-pillow space filled by haloclastite breccia or mudstone (Figure 7.2). Pillow lava is also encountered in outcrop within all pits.

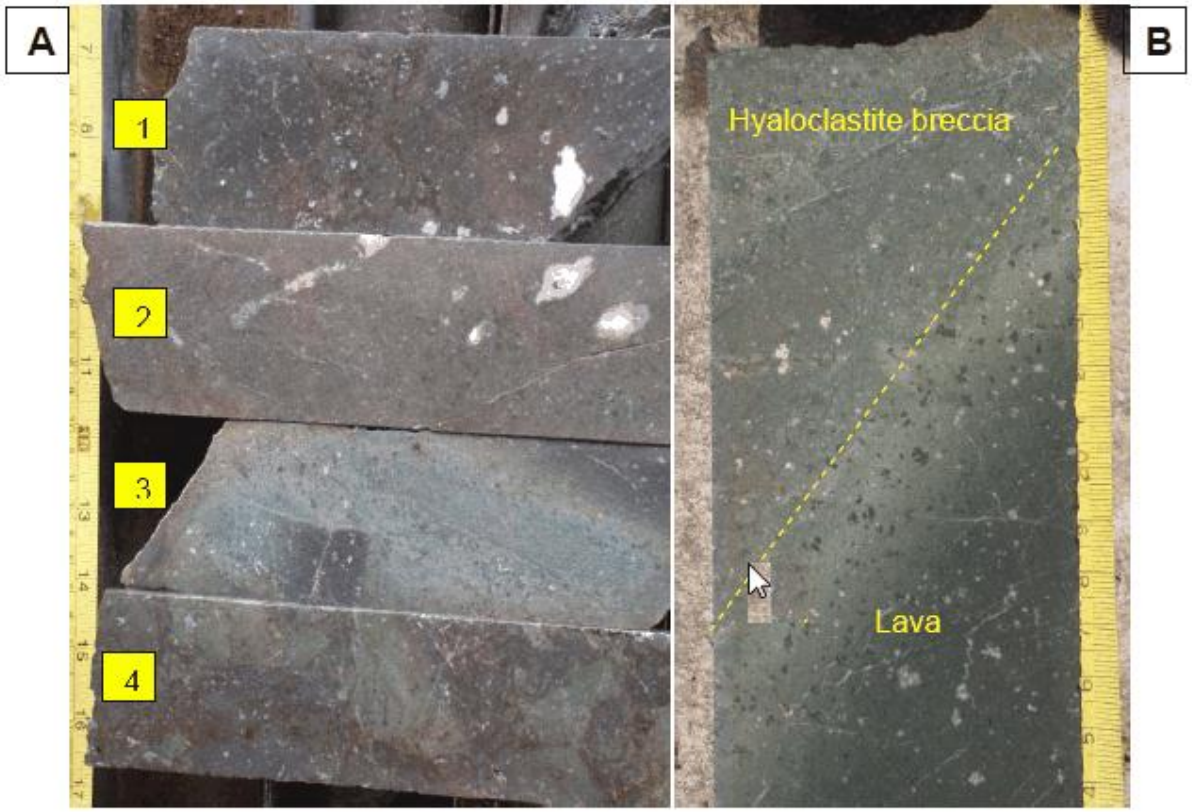


Figure 7.2 *Example of pillowed lava from drill core at MGP. A = Pillowed andesite lava showing progression from an amygdale-rich center of the pillow (i.e. 1 & 2) across a rind (3) into the hyaloclastite breccia margin to the pillow (4). B = Transect across a chilled and 'bleached' rind into hyaloclastite breccia. Sourced from Standing and Halley (2010).*

The presence of extensive hyaloclastite breccia and peperitic breccia, coupled with the pillow structures suggest Masbate andesites were erupted into a deep marine setting.

Conglomerate

Conglomerate is present throughput the mining area, though it is more common in the southeast. Two types of conglomerate are recognised: polymictic and monomictic conglomerate. Polymictic conglomerate is characterised by a number of clast types including basalt, ultramafic, andesite, dacite, felsic porphyry, and a variety of metasediment types (Figure 7.3). Clasts are commonly rounded, indicating they have passed through the fluviatile cycle. Conglomerates range from clast to matrix supported and rarely show any sorting. Narrow sand channels are noted in many locations in pit outcrops. The matrix of the polymictic conglomerate is chloritic, indicating an intermediate to mafic bulk composition.

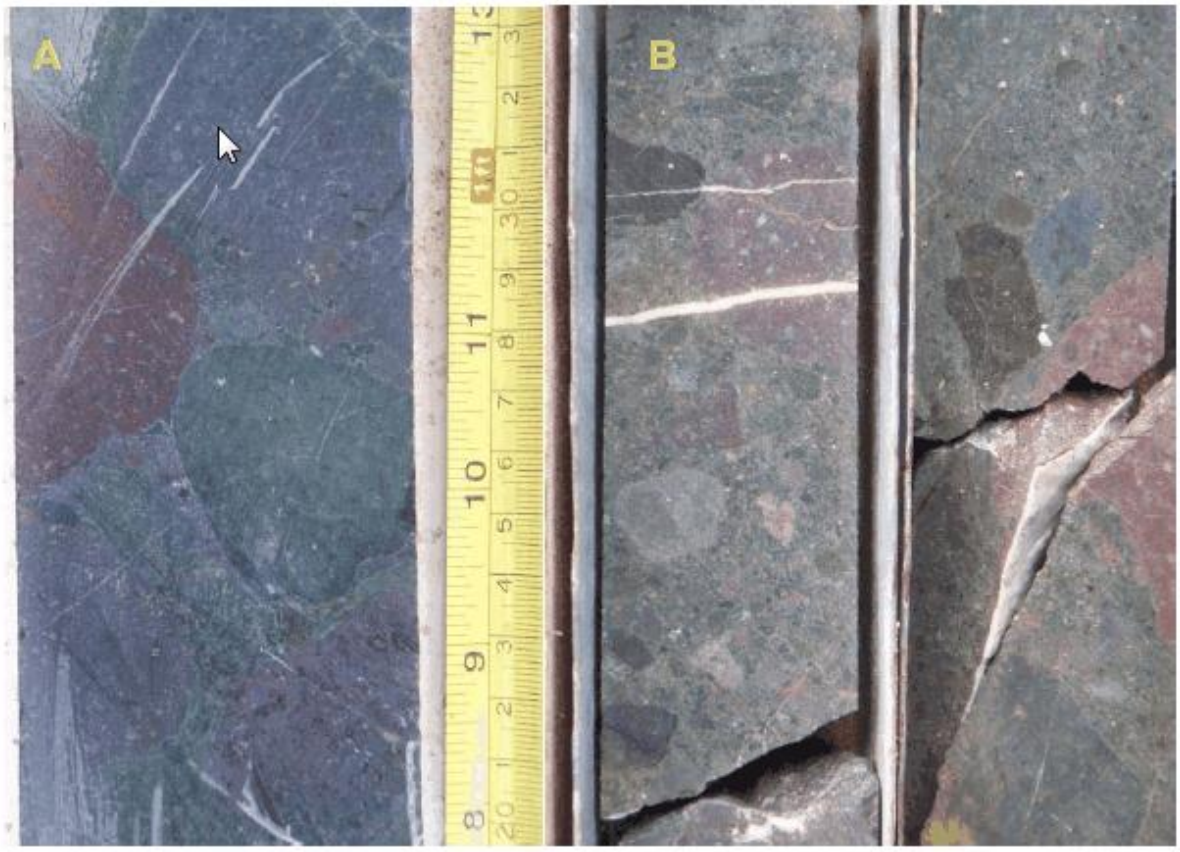


Figure 7.3 *Example of polymictic conglomerate from drill core at MGP. A = Clast-supported cobble conglomerate with well-rounded clasts of various felsic to intermediate volcanics and intrusives. B = Matrix-supported pebble to cobble conglomerate with a multitude of different clast types. Sourced from Standing and Halley (2010).*

Exposures during mining of Libra Pit show polymictic conglomerate regularly contains boulders over 5 m diameter and a number of huge olistoliths of carbonaceous mudstone and sandstone, some over 250 m in length (Figure 7.4). Such rocks are indicative of a submarine canyon setting with catastrophic sector collapse of seabed from the canyon walls into debris flows. The presence of such large blocks creates significant challenges in correctly recognising the true scale and nature of these clasts in drill core.

Turbiditic sandstone and siltstone

Andesitic rocks and conglomerates are interbedded with thin (0.1 to 20 cm) units of feldspathic and variably carbonaceous sandstone, wacke and siltstone. Cross-bedded sandstone, scours, drop stones and shale rip-up clasts are common sedimentary features. These rocks are present as units between cycles of conglomerate as well as olistoliths in polymictic conglomerate and therefore represent both primary depositional and eroded components.

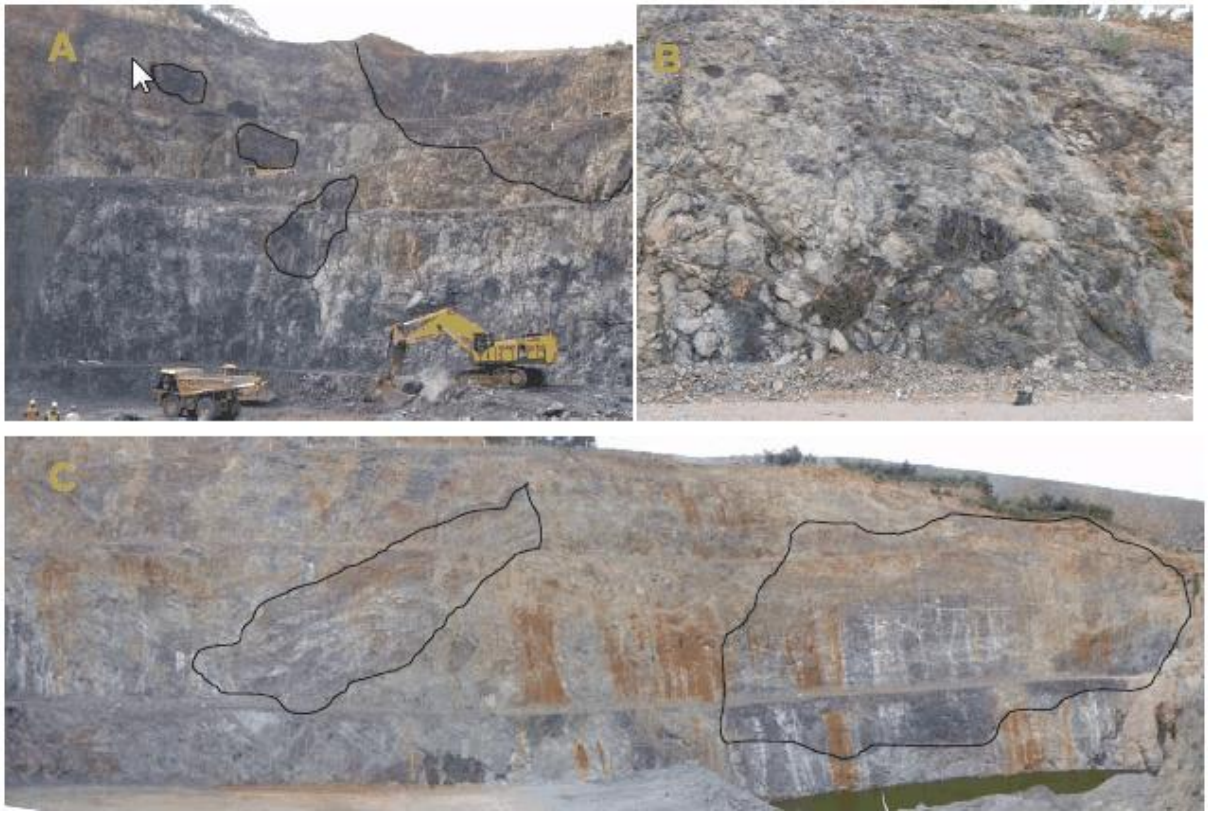


Figure 7.4 *Olistoliths within polymictic conglomerate, Libra West Pit, MGP. A = Panorama of the SE end of Libra West Pit showing a number of large olistoliths of carbonaceous shale. B= Boulders of shale and andesite (wall markings spaced 1 m apart). C = > 250 m long olistolith of carbonaceous siltstone and sandstone encased in polymictic conglomerate, west wall, Libra West Pit. Andesite dykes intrude both the conglomerate and the olistolith. Sourced from Standing and Halley (2010).*

Intrusions

Geological mapping has identified a number of intrusions at MGP and are correlated with the Mt Nabongsoran intrusives of the Masbate Group. A distinctive hornblende – augite porphyry is present as both intrusive bodies and as clasts within polymictic conglomerate. Andesite dykes are observed to intrude conglomerate units in several pit exposures.

7.4.2 Structure

Faulting is the dominant structural style at MGP. Faults are characterised by cataclastic brittle deformation, in particular tectonic brecciation and fault gouge development (Figure 7.5), and fault zones up to 30 to 40 m in width are recognised. Invariably high strain shears bound cataclastic zones and shear surfaces are commonly striated. Outside the cataclastic zones the country rocks are relatively undeformed, with the change from high-strain to unstrained being sharp.

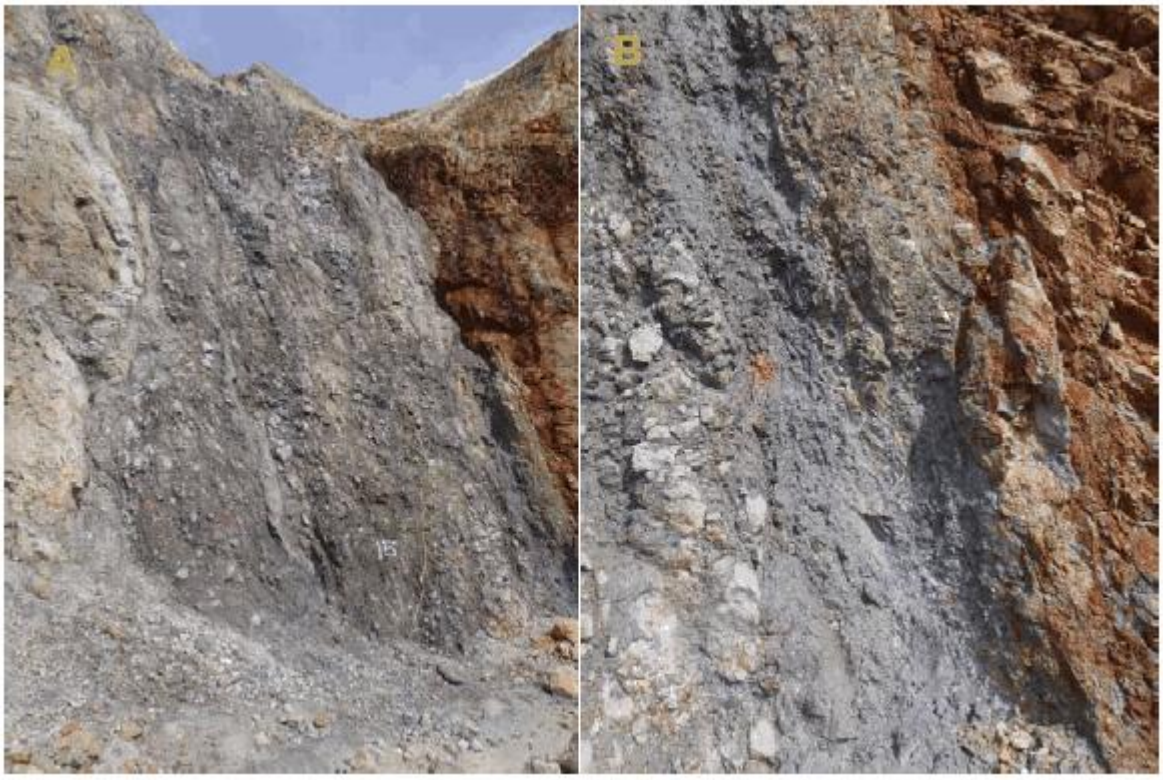


Figure 7.5 *Typical cataclastic fault zone, MGP. A = Libra East Fault – here quartz reef is smashed into a fault breccia of quartz fragments interspersed with puggy fault gouge. B = Close up of high strain and clay rich puggy fault gouge at the margin of the fault zone illustrated in A. Comminution of the quartz reef and wallrock is pervasive. Photo taken at Libra East pit, looking SE. Wall markings are 1 m apart. Sourced from Standing and Halley (2010).*

Faults, puggy shears and shear zones are typically steeply dipping (>65°) and predominantly strike WNW-ESE or NW-SE. Striations and kinematic data record dextral, sinistral and normal movement.

Conflicting or contradicting kinematics highlight the complex and long-lived nature of faulting at Masbate, with the system experiencing sinistral, dextral and normal faulting during its evolution. Sinistral movement is consistent with measured movement along the nearby Philippine Fault and it is expected this is the bulk of movement within the MGP. Sinistral offset of up to 700 m can be inferred along a major cataclastic fault zone linking Montana and Holy Moses – Basalt pits. Ubiquitous dextral kinematics may be a relaxation phase after a sinistral movement event and may not be diagnostic of the bulk movement. Normal movement at shallow crustal levels undergoing extension (orogenic collapse, gravity sliding, back-arc basin opening) is a common feature of the epithermal environment.

7.4.3 Alteration

Alteration is the chemical reaction between hydrothermal fluids and wall rock. Alteration of the mineralised system a MGP has been investigated by spectral analysis of mica and clay minerals, multi-element geochemistry and measurement of magnetic susceptibility.

Multi-element geochemistry indicates there are several magmatic suites represented in the volcanic rocks, predominantly of an andesitic composition.

Alteration associated with mineralisation has stripped all of the sodium and most of the calcium from the original lithologies, and mineralised rock is highly enriched in potassium. Combined geochemistry and spectral analysis shows the alteration associated with mineralisation consist of adularia plus chlorite. Adularia has not previously been reported from MGP and has been widely reported as silicification by previous authors. Microcrystalline adularia associated with chlorite will result in hard, green coloured rocks and are commonly misidentified as silicification. Sericite is also widespread in highly altered samples.

There is a poor correlation between clay minerals and mineralisation, and clay mineralogy is not typical of most epithermal deposits. Mixed illite – smectite is the dominant clay mineral within the MGP, but gypsum is extremely common in Main Vein, Libra and Holy Moses – Basalt, while kaolinite is common at Colorado.

The original clay – mica alteration mineralogy of MGP appears to be zoned around mineralised reefs from illite to illite – chlorite to chlorite (typical of epithermal deposits) but this has been strongly overprinted by a low temperature alteration assemblage. The overprint is pervasive, and particularly strong along mineralised structures.

The presence of thick, barren calcite veins and pervasive smectite – gypsum alteration mineralogy at Main Vein are the result of a collapse of a hydrothermal system which allowed the descent of steam heated water. H_2S from the boiled off vapour has condensed into groundwater, become oxidised to sulphate and them percolated back into the system to form gypsum. Colorado appears to have been located at a shallower level in this collapsing system. At Colorado, the oxidation of H_2S in the absence of significant groundwater has produced sulphuric acid which has reacted with the rock to form kaolinite.

7.4.4 Mineralisation

Gold mineralisation at MGP is hosted by quartz and quartz – calcite veining. Quartz veining is developed in all of the lithologies described above. Individual mineralised veins can be traced up to 3 km; the known system extends over a strike of more than 10 km from Balete in the south to Pajo in the north. The more significant vein networks vary in width from 1 m up to 20 m. At Main Vein, where different structural orientations intersect, a broad zone of alteration or brecciation often occurs resulting in mineralised zones up to 75 m wide. Mineralised vein networks extend to a depth of at least 300 m below the topographic surface – in most instances the total depth of mineralisation has yet to be established.

Drill hole logging and field mapping show a complex evolution of veining with multiple episodes of vein opening that changes over time from quartz and chalcedonic quartz dominant veins to calcite dominant veins (Figure 7.6). Textural evidence indicates high fluid were present in the mineralising system at times during its evolution. Gold is introduced in several stages: in early greyish, sulphide rich quartz, as well as in later sulphide poor chalcedony and calcite veining.

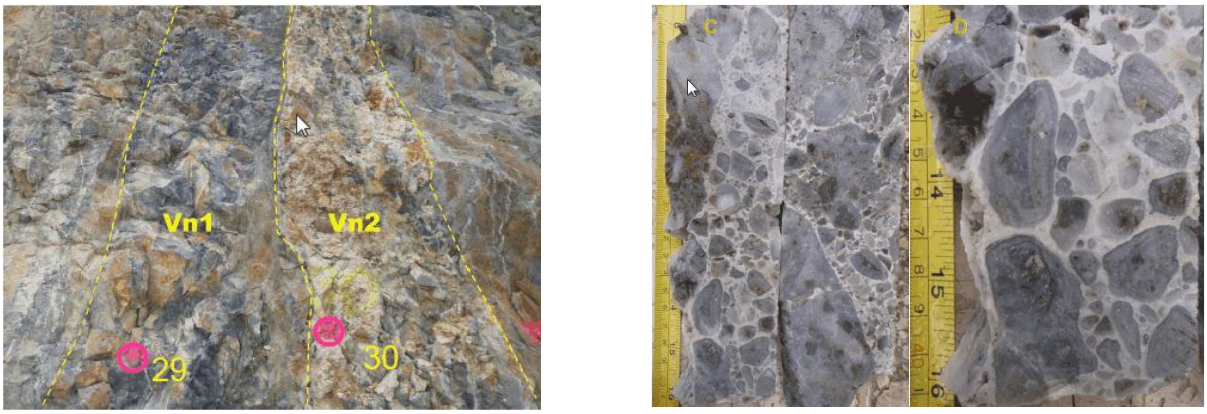


Figure 7.6 *Mineralised quartz, MGP. Left Hand photo = quartz reef at the southern end of Holy Moses – Basalt West Pit showing two stages of evolution; VN1 – greyish, sulphide-rich chalcedonic quartz, overprinted by VN2 – milky white chalcedonic quartz without appreciable sulphide. Photograph taken looking SE. Wall markings (29, 30) are 1 m apart. Right Hand photo = pebble breccia textures from Main Vein. C = two examples of early greyish, sulphide-rich encapsulated by vuggy calcite. D = close-up view of early quartz pebbles showing multi-stage brecciation and cementation prior to encapsulation. Note outer casing of the greyish quartz pebbles are intact and not abraded, suggesting they were suspended in a fluidised bed prior to infiltration of CO_2 rich fluids responsible for calcite deposition. Sourced from Standing and Halley (2010).*

Undeformed mineralised quartz veining is uncommon at MGP. Typically quartz reefs are strongly overprinted by cataclastic deformation and alteration within fault zones (Figure 7.5). Quartz veins predominantly strike NW-SE and that plus textural information indicate that the fault zones now observed at MGP represent renewed brittle deformation of earlier structures that hosted mineralisation. Quartz veins typically dip steeply (>70°) to either the NE or SW.

Fluid inclusions from mineralised quartz veins at Masbate are dominated by two-phase (liquid + vapour) liquid-rich inclusions that have homogenisation temperatures ranging between 219°C and 286°C, with an average of 250°C. The trapped fluids have a low salinity (0 to 3.85 wt. % NaCl equivalent).

Pyrite is the dominant sulphide mineral associated with quartz veining, making up 85 to 95% of all sulphides. Marcasite is locally present. Other sulphide minerals in trace amounts include chalcopyrite, galena and sphalerite. In oxidised rocks, goethite is the dominant ore mineral, with textural evidence suggesting it has formed from the replacement of pyrite.

Gold (more correctly, electrum) is typically observed as < 10 micron inclusions within pyrite or goethite, at the margins of pyrite and other sulphide phases, or more rarely, as free particles up to 500 micron in size (Figure 7.7). Analysis of residual gold in tailings suggests a small amount of gold at Masbate is present within silicate minerals. Elevated Bi and Te contents in mineralised samples indicate gold telluride minerals may also be present, although these have been observed.

The silver (Ag) content of Masbate ore is variable but is typically present in a ratio of 0.5 – 2 times the gold content. Ag content is not routinely analysed during exploration and grade control activities and is not estimated in resource and reserve calculations. Dore produced by the Masbate Process plant typically contains 40 to 45 % Ag.

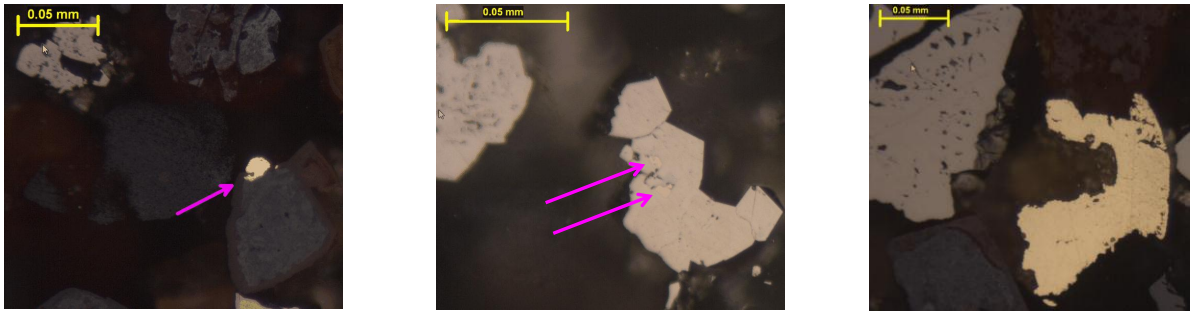


Figure 7.7 *Gold mineralogy in polished sections of cyclone overflow samples collected from the Masbate Processing plant, MGP. Left Hand photo = gold inclusion in goethite. Central photo = two gold inclusions in pyrite. Right Hand photo = free particle of gold. Scale bar in all photos equates to 50 microns. All photos sourced from Townend (2011).*

8 DEPOSIT TYPE

8.1 General

The MGP is a low sulphidation epithermal gold deposit.

Epithermal deposits are a widely recognised type and generally well understood type of ore deposit and have received considerable academic, mineral exploration and mining interest (Corbett & Leach 1997, Simmons et al 2005 and references therein). Intrusions emplaced in volcanic arcs provide heat sources to drive deep circulating hydrothermal systems which are composed of varying proportions of magmatic and meteoric fluids. Magmatic volatiles which evolve from the cooling magma (H_2S, CO_2, HCl) become entrained at the base of the circulating fluids. Epithermal deposits represent zones of permeability within the shallow parts (generally < 2 km from the surface) of these hydrothermal systems (Figure 8.1).

Epithermal deposits are generally hosted by coeval and older volcanic rocks and/or underlying basement rocks. They cover areas that range from <10 to >100 km^2 and ore bodies occur in a diversity of shapes that reflect the influence of structural and lithological controls. Most commonly, ore bodies occur in veins with steep dips that formed through dilation and extension. Some are hosted by major faults but more commonly they are hosted by minor faults (second- or third-order structures) with small displacements. Optimum structural development generally depends on rock rheology and brittle failure. Lithology is also important, especially where contrasts in porosity and permeability focus fluid flow through specific units, along rock contacts, or through permeable masses of brecciated rock. These lithologic features may be an intrinsic characteristic of the original rock or they may be a by-product of hydrothermal alteration and chemical dissolution or hydrothermal brecciation. Thus, faults and fracture networks, as well as breccias, coarse clastic rocks, and intensely leached rocks account for the spectrum of vein-related to disseminated ores, which can extend for 100's to 1,000's of meters laterally and 10's to 100's of meters vertically. The dominant gangue mineral is quartz, making ores hard and generally resistant to weathering, and the dominant sulphide mineral is pyrite, with sulphide contents that can range from <1 to >20 volume %.

A number of sub classifications have been identified within epithermal deposits (Figure 8.1).

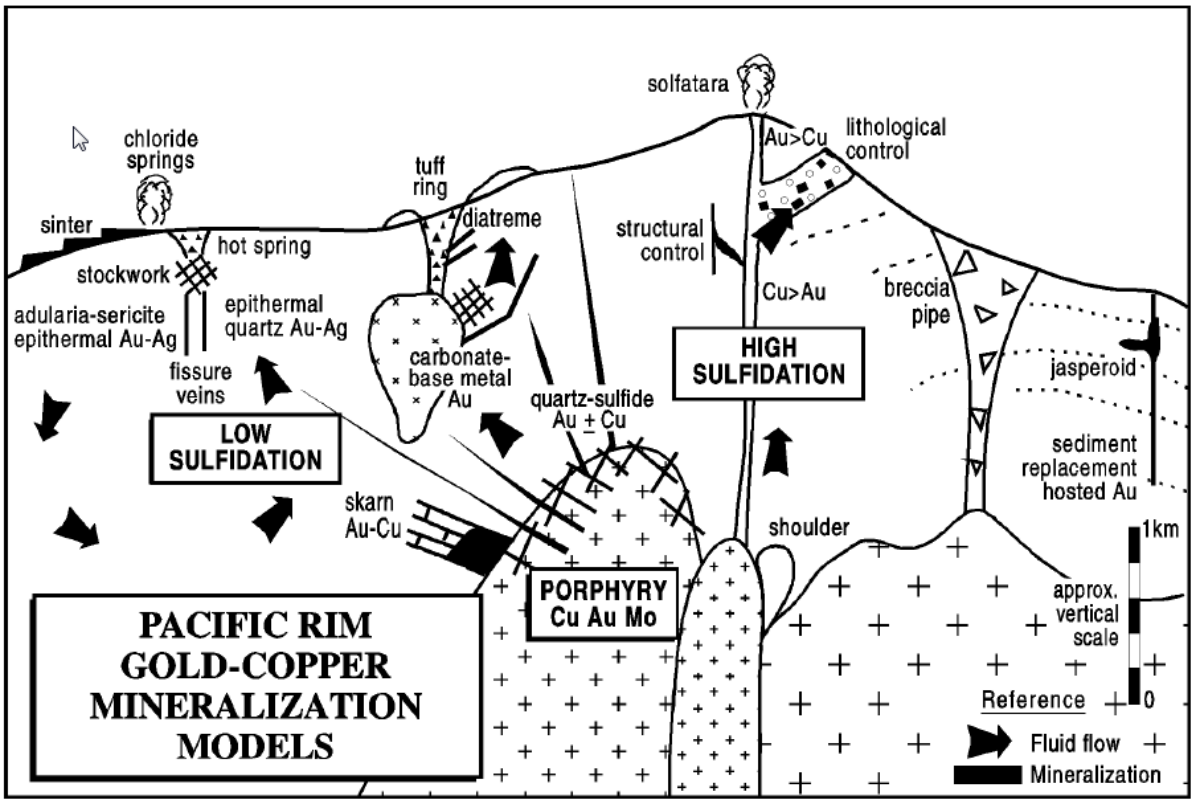


Figure 8.1 *Schematic model of epithermal deposits and their relationship to porphyry systems (Corbett & Leach 1997).*

8.2 Low Sulphidation Epithermal Deposit

Low sulphidation epithermal Au-Ag deposits are distinguished from high sulphidation deposits primary by different sulphide mineralogy (pyrite, chalcopyrite, galena, sphalerite), typically present within quartz veins with local carbonate. Clay mineralogy formed during alteration associated with mineralisation (generally dominated by illite) indicates low sulphidation deposits are deposited from near neutral fluids that have a high meteoric component. Fluid inclusion studies indicate fluids are dilute (< 5 wt. % NaCl equivalent) with temperatures between 150°C to 300°C. Textural and fluid inclusion evidence boiling of fluids occurs at time during ore deposition. Ore occurs in veins and stockworks, generally in sub vertical fractures.

Portions of the MGP currently being mined formed in low suphidation epithermal system where fluid permeability was created, and ore deposition occurred within, a network of brittle faults. Although dating of mineralisation is not yet available, fault development and dilution is likely to have been initiated by early movement along the nearby Philippine Fault. Continued movement along the Philippine Fault has reactivated the subsidiary network work of faults, in many cases deforming mineralised quartz reefs. Reactivation of the faults also allowed the migration of low temperature fluids that produced a mineralogy that has partially overprinted the original epithermal alteration assemblage.

9 EXPLORATION

9.1 General

Mining and processing operations at MGP occupy approximately 20% of the current total tenement area. Several smaller mineralised quartz deposits occur within close proximity to the current mining operations and are the subject of continued exploration. Elsewhere in the Philippines (e.g. Far SE Deposit, Lepanto), low sulphidation epithermal deposits are spatially associated with other types of mineralisation (High sulphidation epithermal Au-Ag, Porphyry Cu-Au) and potential exists for additional mineralisation within the MGP tenement holding.

9.2 Previous Exploration

Exploration activities by previous companies (Atlas, 1979 to 1994; Base Metals, 1997 to 1998; FRC 2001 to 2006) have generally been confined to the area of current mining operations. Significant results are documented in Section 6.3 and 6.4. Exploration activities included field mapping, trench sampling, sampling and mapping of underground workings, and database compilation, in addition to auger, RC and diamond drilling.

Limited exploration has been conducted by previous companies outside of current mining areas. Atlas conducted some mapping and sampling of stream sediments. Several prospects within a 25 km radius of the Atlas mining operation were identified but subjected to limited exploration.

Atlas conducted exploration of the Pajo deposit under agreement within the then tenement holder during 1990 to 1992. Atlas mapped and sampled the surface and old underground workings, and drilled 15 holes, but did not commit to a systematic drill programme to assess the resource.

9.3 Current Exploration

Following completion of construction and commissioning of MGP operations during 2009 to 2010, FRC has embarked upon a significant exploration program with the goal of materially enhancing the reserve and resource base of the project. In April 2010 FRC was granted a 52 km^2 Exploration Permit adjacent to south and east of the mining areas (EP-010-2010-V) and from December 2010 gained access to a further 786 ha located immediately to the north (MPSA-219-2005-V). Both permit areas include a number of known and previously mined gold deposits as well as hosting a number of geophysical anomalies. Both permits have not been previously explored with modern or systematic exploration.

9.4 Regional Mapping and Surface Sampling

Early in February of 2011 Filipino geological contractors, GTS, were engaged to carry out detailed field mapping, stream sediment sampling, and rock chip sampling over EP-010-2010-V and Vicar MPSA-219-2005-V. By the end of the reporting period the team of 4 geologists and 14 field assistants had covered 50 % of EP-010 and 100% of MPSA 219.

Stream samples were taken at a density of 50 / km^2. Sample sites were selected depending on drainage type/size, from 1st-order to 3rd order streams. Typically 3 samples are collected at stream junctions - one each for the 2 upper drainages and another one for the higher order drainage downstream. Two samples were taken at each location, a wet sieved -80# sample of 1.5 kg and a

heavy mineral concentrate of 300 – 350 g. Heavy mineral concentrates were examined under hand lens and any visible gold recorded.

Rock chip samples were taken during the course of mapping activities, in areas of particular interest, channel samples were taken across mineralised outcrops. Details of the number and type of samples collected are presented in Table 9.1 and sample localities are illustrated in Figure 9.1.

Tenement	-80# stream	HM Stream	Rock chip
EP-010-2010-V	545	548	968
MPSA-219-2005-V	45	45	158

Table 9.1 *Stream sediment and rock chip sampling, 2011 MGP regional exploration. HM = heavy mineral.*

The -80# stream samples and rock chip samples were assayed for Au using 50g Fire Assay followed by ICPAES finish to give upper and lower detection limits of 1 – 10,000 ppb. They were also analysed for a multi-element suite by ICP (Table 9.2).

Element	Detection Limit	Upper Reporting level	Element	Detection Limit	Upper Reporting level
Ag	2 ppm	10 ppm	Mn	2 ppm	1%
Al	0.01%	15%	Mo	1 ppm	1%
As	3 ppm	1%	Na	0.01%	15%
Ba	1 ppm	1%	Ni	1 ppm	1%
Be	0.5 ppm	0.25%	P	0.01%	15%
Bi	5 ppm	1%	Pb	2 ppm	1%
Ca	0.01%	15%	S	0.01%	5%
Cd	1 ppm	1%	Sb	5 ppm	1%
Co	1ppm	1%	Sc	0.5 ppm	1%
Cr	1ppm	1%	Sn	10 ppm	1%
Cu	0.5 ppm	1%	Sr	0.5 ppm	5%
Fe	0.01%	15%	Ti	0.01%	15%
Hg	1 ppm	1%	V	2 ppm	1%
K	0.01%	15%	W	10 ppm	1%
La	0.5 ppm	1%	Y	0.5 ppm	1%
Li	1 ppm	1%	Zn	0.5 ppm	1%
Mg	0.01%	15%	Zr	0.5 ppm	1%

Table 9.2 *ICP multi-element suite, stream sediment (-80# mesh) and rock chip samples, 2011 MGP regional exploration.*

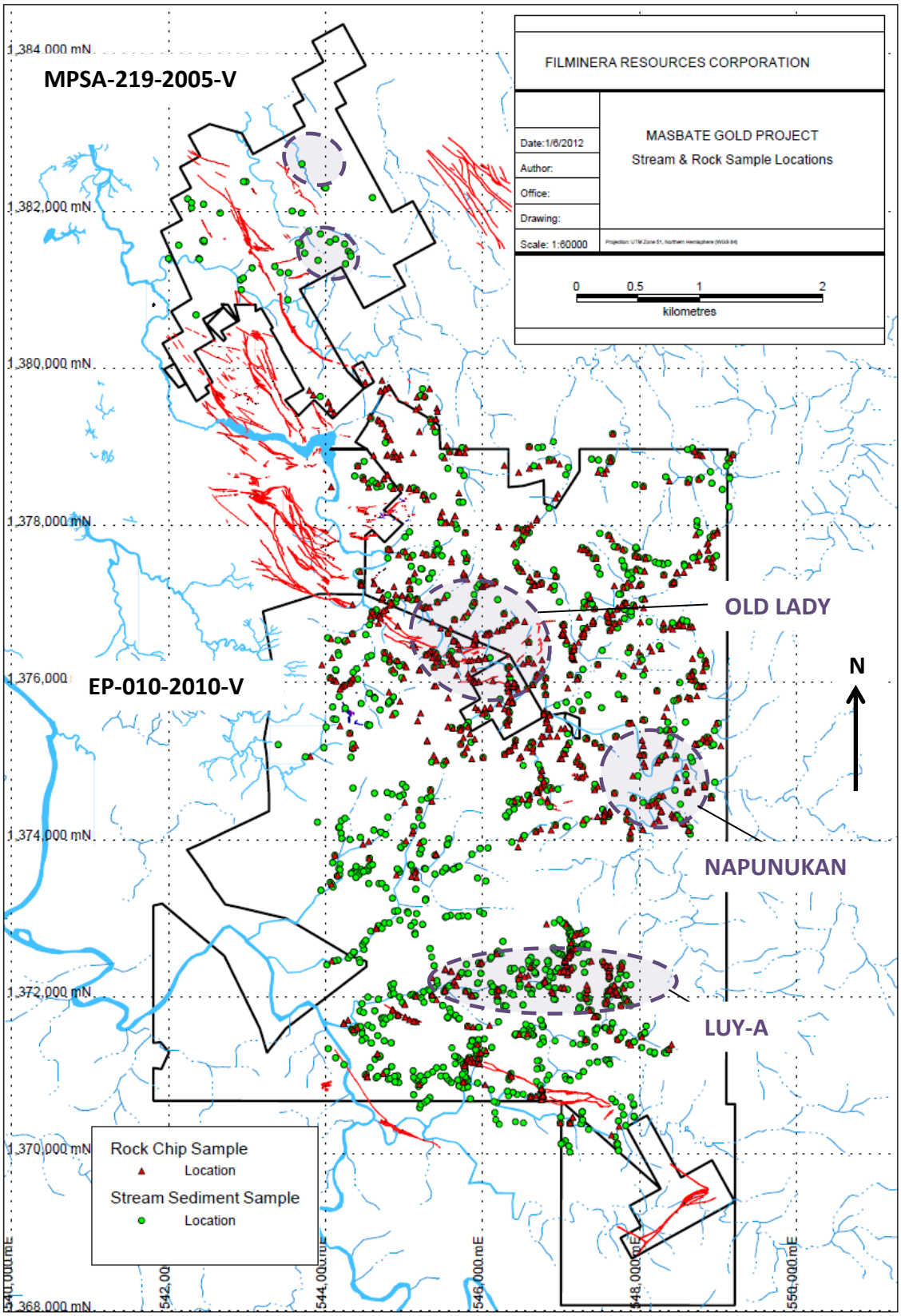


Figure 9.1 *Rock chip and stream sediment sample locations, 2011 MGP regional exploration. Red lines illustrate the location of known veins. Shaded areas illustrate anomalous areas from results to date: 3 areas in EP-010-2010-V (Luy-A, Napunukan and Old Lady) and 2 unnamed areas in MPSA-219-2005-V.*

Stream sediment and rock chip sampling are used in mineral exploration to identify drainage catchments and other areas that may host mineralised deposits and are not representative of the size and grade tenor of any such deposits. Results received from Au analysis of stream and rock chip sampling to date define three co-incident gold in drainage / rock chip anomalies in EP-010-2010-V (Figure 9.1). The northern most anomaly is associated with outcropping veins in the Old Lady – Water West area, the anomaly to the SE may be associated with the Napunukan veins, and the southern anomaly is associated with the Luy-A vein. Two unnamed anomalous areas are defined from stream sediment samples in MPSA-219-2005-V.

The multi-element suite is currently being analysed to determine if other anomalies and target types are present within the survey area.

9.5 Geophysics

A helicopter-borne magnetics and radiometric survey was conducted within EP-010-2010-V during 2009. The total survey length was 996.8 line km. Processing, modelling and interpretation of the data indicate that known MGP mineralisation (Main Vein, Colorado, etc.) has a direct radiometric (potassium) and magnetic response.

Imaging of radiometric data (Figure 9.2) shows a strong potassium anomaly over the current mining area of Main Vein and a similar signature 4 km to the east at Pinanaan. Isolated potassium anomalies are also present to the south, at David Sun and Art-Ag / Balete prospects.

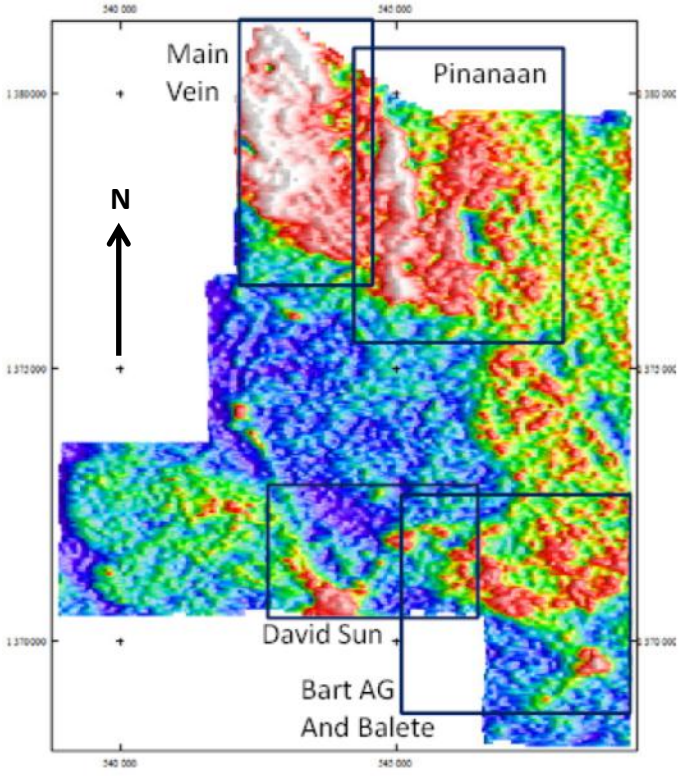


Figure 9.2 *Radiometric signature for potassium, MGP. Anomalous areas (red and white) are present over known mineralisation (and current mining areas) in Main Vein. A similar anomalous area is located 4 km to the east at Pinanaan. Isolated anomalies are also present to the south at David Sun and Bart-Ag / Balete prospects. Processed data collected from airborne survey completed during 2009.*

Processing and interpretation of magnetic data highlights a series of magnetic lows associated with magnetite destruction as a result of hydrothermal alteration (Figure 9.3). This approach has defined most areas of outcropping mineralisation and highlighted additional targets for testing. In many examples of known mineralisation, magnetic lows are located along the margins of magnetic highs interpreted as unaltered rocks of andesitic composition.

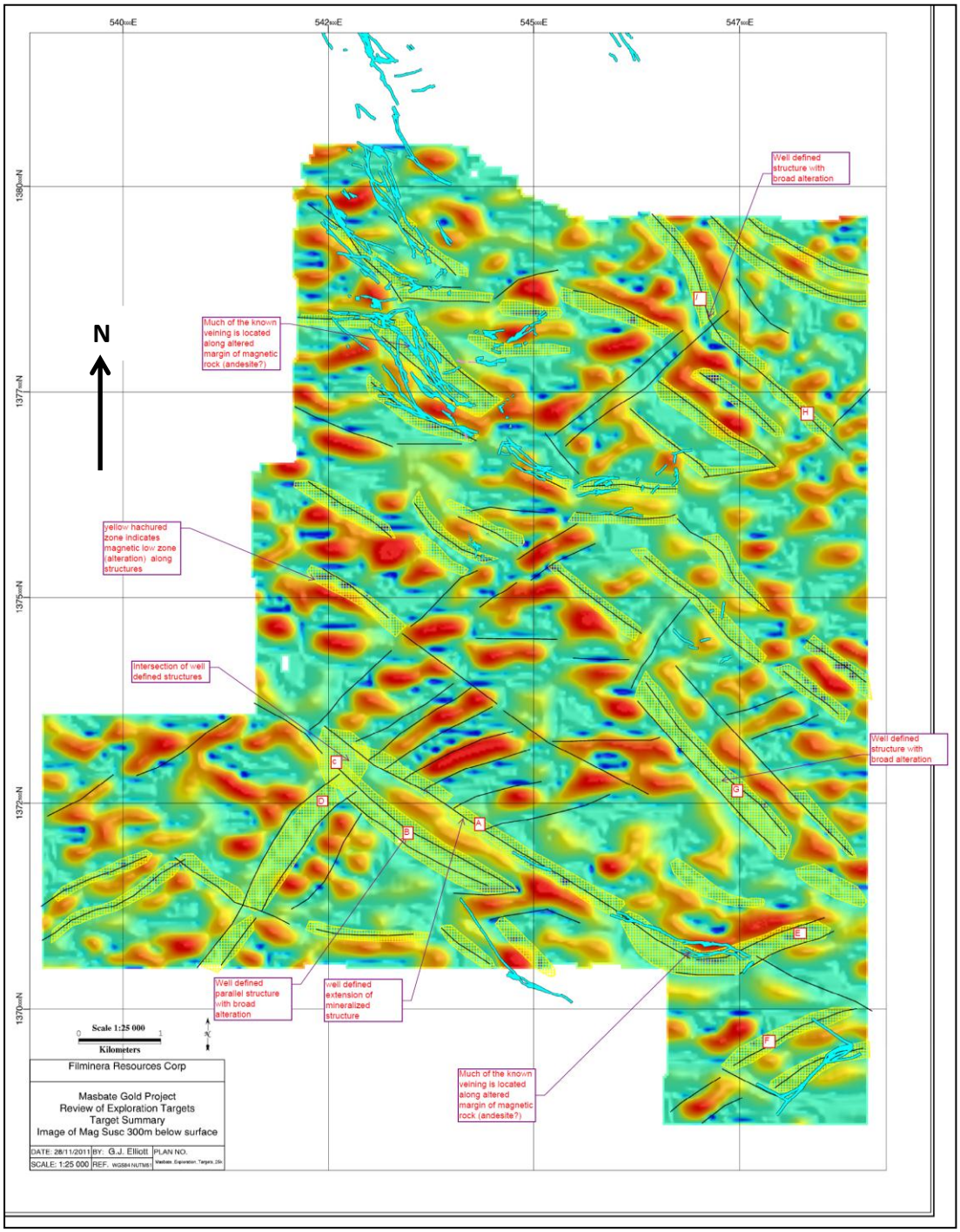


Figure 9.3 *Inverted magnetic signature, 300 m below surface, MGP. Low magnetic signature (blue / green) interpreted as a result of magnetite depletion through alteration (yellow cross hatch). This interpretation has defined most of the known outcropping mineralized quartz veins and highlights several targets for further testing. Processed data collected from airborne survey completed during 2009.*

An orientation Induced Polarisation (IP) survey was conducted across and area of known mineralisation (Blue Quartz, Boston, Porphyry vein sets) in 2011 to test the effectiveness of this technique. Resistivity successfully delineated known mineralisation and identified possible extensions of these zones beneath areas of shallow soil cover (Figure 9.4).

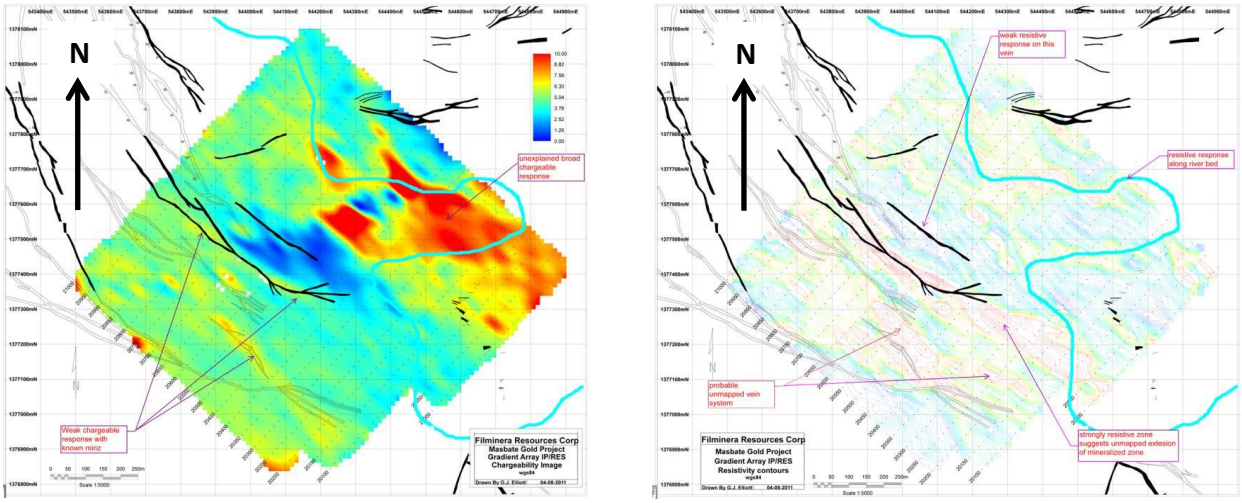


Figure 9.4 *Resistivity (left) and chargeability (right) for an orientation IP survey conducted over known mineralization, 2011, MGP. Known mineralization (open black lines and heavy black lines) generally have a weak resistivity response (yellow) but a strong chargeability response (pink). Chargeability response suggest known veins extend beyond mapped limits and requires follow up drill testing. (CGA 2011)*

9.6 Exploration Drilling

The bulk of drilling conducted from late 2009 to the present date has been within the area of updated mineral resource / mineral reserve and is discussed in section 10 of this report.

Two diamond drill holes have been completed at Pinanaan, targeting the central portion of the potassium anomaly described in section 9.3.2. Both drill holes collared in diorite and intersected minor amounts of narrow quartz veining with low level gold and base metal values (Table 9.3).

Drill Hole ID	EOH	Dip	From	To	DH Length	T Length	Grade
PINDH001	350.5	50°	120	121	1	0.5	0.76
PINDH002	354	55°	38	39	1	0.5	0.68
			76.1	76.9	0.8	0.5	2.97

Table 9.3 *Significant results from diamond drilling of Pinanaan deposit, 2010. Significant interval defined as ≥ 1 m interval with Au grade ≥ 0.4 g/t, no more than 2 m intervals of internal waste (grade > 0.25 g/t), no top cut. ID = Identity, DH length = drill hole length, T length = true length (assuming vertical dip).*

10 DRILLING

10.1 Historical Drilling

Atlas, Base Metal and FRC drilling programs for the period 1975 to 2008 are described in detail within the 2006 and 2008 MA NI43-101 reports. Some 120,462 m of drilling was conducted by various companies during this period (Table 10.1).

Year	Company	RCDH		DDH		Total	
		holes	***metres***	***holes***	***metres***	***holes***	***metres***
1975-98	various	377	30,156	532	58,043	909	88,199
2002-2005	FRC	326	25,351	6	714	332	26,065
2007 -2008	FRC	108	6,198			108	6,198
Total FRC		*434*	*31,549*	*6*	*714*	*440*	*32,263*
Total		**811**	**61,705**	**538**	**58,757**	**1,349**	**120,462**

Table 10.1 *Summary of historical drilling within area of MGP resource estimate.*

Atlas drilling (1975 – 1994) was generally conducted using company-owned equipment for both reverse circulation (RC) and diamond drilling. Much of the diamond drilling was conducted from within existing underground workings or during the course of Atlas underground mining operations.

RC drilling was confined to relatively shallow depths and specific open pit prospects. Most RC drilling was carried out using 4 ½ inch diameter drill bits and cross-over subs. Maximum depths were 100 m, with most drill holes averaging 50 – 60 m. The majority of samples were dry.

Drilling by Base Metals (1997 – 1998) focused on six prospects where Atlas had records had indicated significant potential – Main Vein, Libra, Holy Moses – Basalt, Colorado, Grand View and Montana. Approximately 90 diamond drill holes were completed to confirm Atlas results, gather geologic information and test deeper targets. Approximately 165 RC drill holes provide infill information and tested near surface potential. Drill spacing varied between prospects with an average 30 m sections along strike and 50 m across strike, with down hole depths between 50 to 100 m. In Main Vein pit, systematic coverage of some areas was difficult due to access problems related to the location of Atlas pit walls.

Diamond drilling was completed by Exploration Drilling Corporation (EDCO), a Canadian owned drilling company. Diamond drill holes were started with PQ core size, changing to HQ in competent rock. A few of the deepest drill holes reduced to NQ core size. RC drilling was conducted by East West Drilling, an Australian drilling contractor, using 5 ¼ inch diameter bits and face sampling hammers. Behre Dolbear Australia (BDA) visited the site several times during drilling by Base Metals and considered the drilling programme to be conducted competently and a sound basis for evaluation of the deposits.

FRC (funded by Thistle Mining) drilling during 2002 to 2004 was completed by East West Drilling (RC) and EDCO (diamond) drilling contractors using similar equipment and methods to previous campaigns. The drilling contractors were replaced by United Philippines Drilling in 2005, who conducted a further diamond and RC drill programme.

FRC (funded by CGA) drilling during 2007 and 2008 was completed by DrillCorp Asia (RC and diamond) drilling contractors. RC drilling was conducted using 5 ¼ inch diameter bits and face sampling hammers. Diamond drill holes were started using PQ core size, changing to HQ in

competent rock. A few of the deepest drill holes reduced to NQ core size. In some instances RC drill holes were completed using HQ (and subsequently NQ if required) diamond core size.

10.2 2009 to 2011 Drilling

10.2.1 Exploration / Resource Drilling

Exploration and / or resource drilling completed during 2009 to present is conducted by DrillCorp (RC and diamond) and Bradley Drilling Incorporated (diamond) drilling contractors. DrillCorp equipment and drilling methods are similar to that used in the 2007 to 2008 drilling campaign. Bradley drilling uses PQ, HQ and NQ diameter core size.

A summary of drilling programmes and prospects completed during the period December 2009 to August 2011 are presented in Table 10.2 and drill hole collars illustrated in Figure 10.1.

Prospect	DDH		RCDH		Total	
	holes	***metres***	***holes***	***metres***	***holes***	***metres***
Grand View			18	2,114	18	2,114
Colorado	1	200	7	1,280	8	1,480
Holy Moses – Basalt	3	555	35	4,256	38	4,811
Libra	12	1,677	16	2,738	28	4,416
Binstar	16	5,089	22	2,665	38	7,754
Main Vein	1	311	18	2,906	19	3,217
Boston			90	10,200	90	10,200
Blue Quartz			35	4,516	35	4,516
Old Lady			28	3,358	28	3,358
TOTAL	**33**	**7,832**	**269**	**34,034**	**302**	**41,866**
Geotechnical	14	3,624			14	3,624
Sterilisation – Colorado			146	11,572	146	11,572
Sterilisation – Holy Moses			24	1,090	24	1,090
Sterilisation – Boston / Lacer			80	7,938	80	7,938
TOTAL	**14**	**3,624**	**250**	**20,600**	**264**	**24,224**

Table 10.2 *Summary of 2009 – August 2011 drilling by prospect or programme conducted within the area of 2011 MGP mineral resource estimate.*

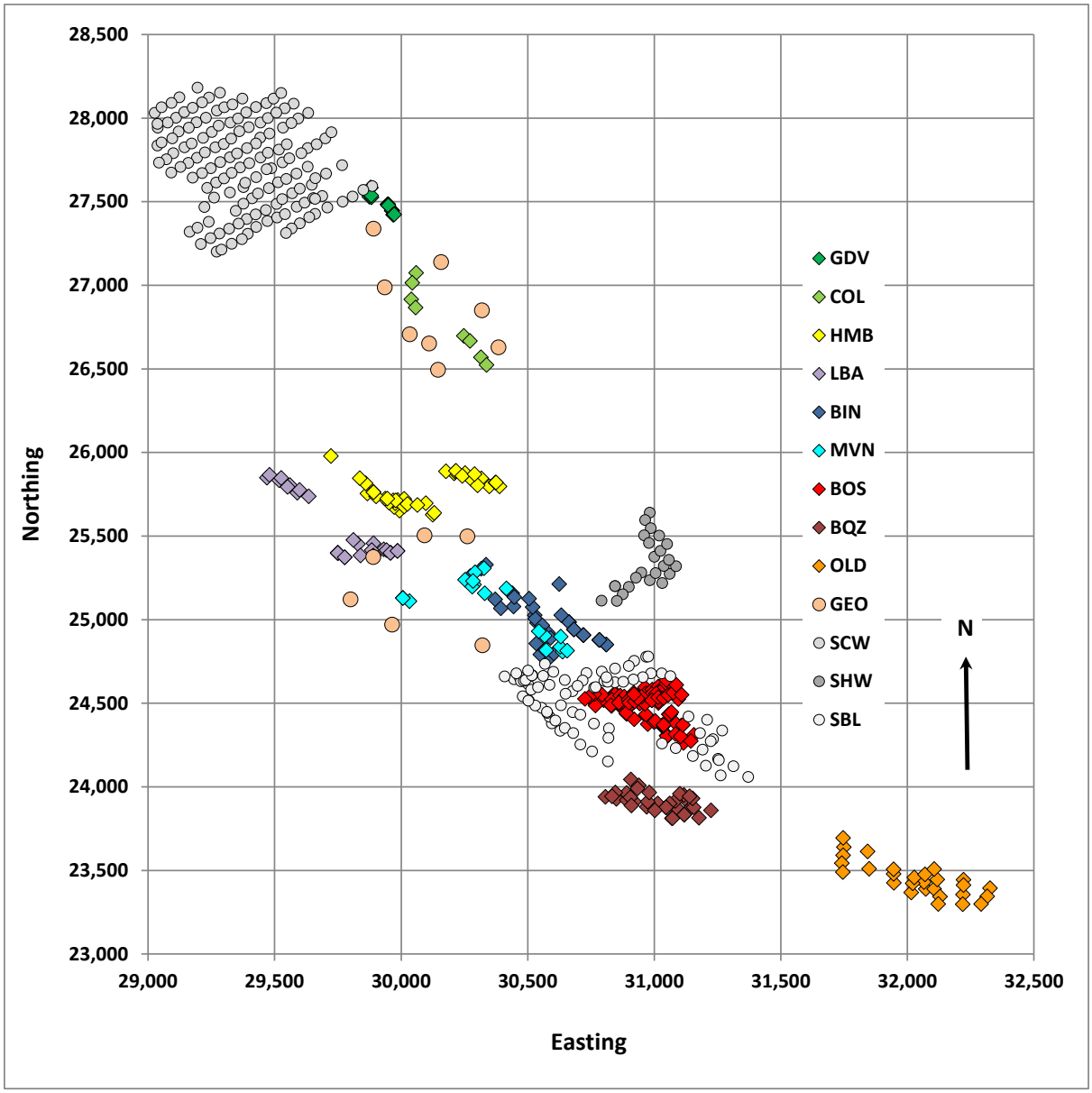


Figure 10.1 *Location of Exploration / Resource drilling from 2009 utilised within the MGP 2011 mineral resource estimate. GDV = Grand View prospect; COL = Colorado prospect; HMB = Holy Moses – Basalt prospect; LBA = Libra prospect; BIN = Binstar prospect; MVN = Main Vein prospect; BOS = Boston – Porphyry prospect; BQZ = Blue Quartz prospect; OLD = Old Lady prospect; GEO = Geotechnical drill programme; SCW = Colorado Waste Dump sterilization drill programme; SHW = Holy Moses Waste Dump extension sterilization drill programme; SBL = Boston – Lacer sterilization drill programme. (CGA 2012)*

10.2.2 Grade Control drilling

Grade control drilling at MGP is conducted by dedicated RC drill rigs operated by DrillCorp using 5 ¼ inch diameter bits and face centred hammers. Up to May 2010, grade control drilling was conducted from surface, on a nominal spacing of 12 m by 12 m and a variable drill hole length (average 40 to 60 m, maximum lengths > 100 m). Drill hole collars were surveyed prior to and after hole completion and down hole surveys complete every 50 m and / or end of hole (EOH). Grade control drill holes utilising these drilling procedures are identified by the prefix "RCGC" within the drill hole database.

From May 2010, grade control drilling has been incorporated within the mining cycle and standardised to a 6 m cross strike by 12 m along strike drill spacing and 12 m down hole length (equating to 10 m vertical). Drill hole collars are surveyed prior to and after drilling. There is no down hole survey. Grade control drill holes utilising these drilling procedures are identified by the prefix "GCH" within the drill hole database. At current mining rates, grade control drilling requirements are > 160,000 m per annum.

10.3 Drill Hole Survey

10.3.1 Collar Survey

The collar location of Atlas drill holes were surveyed prior to drilling, with position plotted on cross section along with the dip of the hole. No post-drilling systematic checking of collar location or down hole azimuth and dip was carried out.

During database review, BDA found no survey records to enable checks on Atlas drill hole locations and orientations as plotted on geologic maps. The database of accepted locations for Atlas drilling was reassessed, validated and corrected in several stages by Base Metals (1997, 1998), BDA (1997, 1998) and FRC (2006) in response to irregularities between Atlas sections and plans.

Drilling conducted by Base Metals was systematically surveyed before and after drilling.

Drilling conducted by FRC during 2002 – 2005 and 2007 – 2008 were surveyed prior to and after drilling, with drill hole collars capped after drilling by a named concrete plug. Collars were surveyed via theodolite or total station from known base stations, the position of which was confirmed by independent surveyors.

Exploration and resource drilling completed since 2009 are surveyed prior to and after drilling, with drill hole collars capped after drilling by a named concrete plug. In active mining areas, collar locations prior to drilling are set out by mine-based survey personnel by total station or DGPS. In more remote locations, collars may be set out using hand held GPS. The collars of all completed drill holes are picked up by mine-based survey personnel using total station or DGPS utilising known and regularly checked base station locations.

10.3.2 Down Hole Survey

Atlas drill holes were generally not survey down hole.

The majority of Base Metal diamond drill holes were surveyed down hole at nominally 50 m intervals and EOH using a clockwork Topari instrument. Down hole surveys of RC drilling was not carried out as holes were generally less than 100 m in length. Drill hole path was determined from the orientation and inclination at the collar. BDA (1998) concluded that Base Metal and Atlas drilling collar and down hole surveys were adequate for resource estimation.

RC and diamond drilling completed by FRC during 2002 to 2004 was surveyed through stainless steel rods using an Ausmine battery operated, single shot camera at nominally 50 m intervals.

All RC drilling completed in 2005 were surveyed through a stainless steel rod, with diamond drill holes surveyed using a brass camera spear extended out through the drill bit. One Ausmine and one

Eastman battery operated, single shot camera was used to survey each hole at nominally 50 m intervals. Some down hole surveys produced inconsistent results, later identified to be related to the amount of magnetite in unaltered rock. In affected drill holes, measured down hole dip was retained but the collar azimuth survey was substituted for all down hole camera readings.

FRC drill holes during 2006 to 2008 were surveyed down hole through stainless steel rods using an Eastman battery operated, single shot camera. Each hole was surveyed at nominally 60 m intervals and EOH. No significant deviation issues were identified.

All RC drill holes completed during 2009 to 2011 were surveyed through a stainless steel rod. All diamond drill holes are surveyed using a brass camera spear extended out through the drill bit. Down hole survey instruments used are Proshot and Reflex. Surveys are taken every 50 – 60 meters, as the hole is being drilled, to ensure down hole deviations in the dip (sag) or the bearing of the hole is within acceptable limits (3° per 100 m). When holes start deviating more than acceptable, drillers are informed, and "monitoring surveys" done every 25 - 30 m. Holes which deviate unacceptably are abandoned, measures are taken to find out the cause and precautions are implemented for the next holes.

Anomalous down hole survey magnetism has been minimal. All readings with anomalous nT values are resurveyed to ensure magnetism is not caused any driller or equipment error. In cases where the anomalous magnetism reading is a result of lithology (amount of magnetite in unaltered rock), down hole dip was retained but the collar azimuth survey was substituted for the down hole camera readings.

10.4 Sample Recovery

Detailed records of FRC drilling recovery versus gold grade have been documented from FRC drilling completed during 2005 (Table 10.3).

Au Grade (g/t)	HQ Recovery (%)	RC Recovery (%)
0.05 to 0.50	96.3	83.8
0.50 to 1.20	87.2	70.0
Above 1.20	81.3	76.0
All Data	93.7	80.9

Table 10.3 *Drilling recovery versus gold grade, 2005 Base Metal drilling campaign. HQ = diamond drill core with diameter of 63.5 mm. Sourced from Vigar (2006)*

Overall diamond core recovery was 93.7%, but a trend of decreasing core recovery with increasing grade of mineralisation is apparent. A similar trend is present in RC drilling recoveries. Overall, RC drill recovery averaged 80 %. As noted in Section 7, mineralised quartz reefs have often been severely deformed by fault movement subsequent to vein formation (Figure 7.5) and reduced drill recoveries from broken and highly fractured intervals is to be expected. MA considered that despite the noted decrease in recoveries through mineralised zones, drill recovery was satisfactory.

Figure 10.2 illustrates a more complex relationship between diamond drill core recovery and gold mineralisation from Drill Hole BMVDH014, drilled during July 2011. Some zones of gold mineralisation are associated with zones of reduced diamond core recovery, but the same drill holes

also includes mineralised zones with excellent recovery and a zone of poor recovery that is not associated with mineralisation.

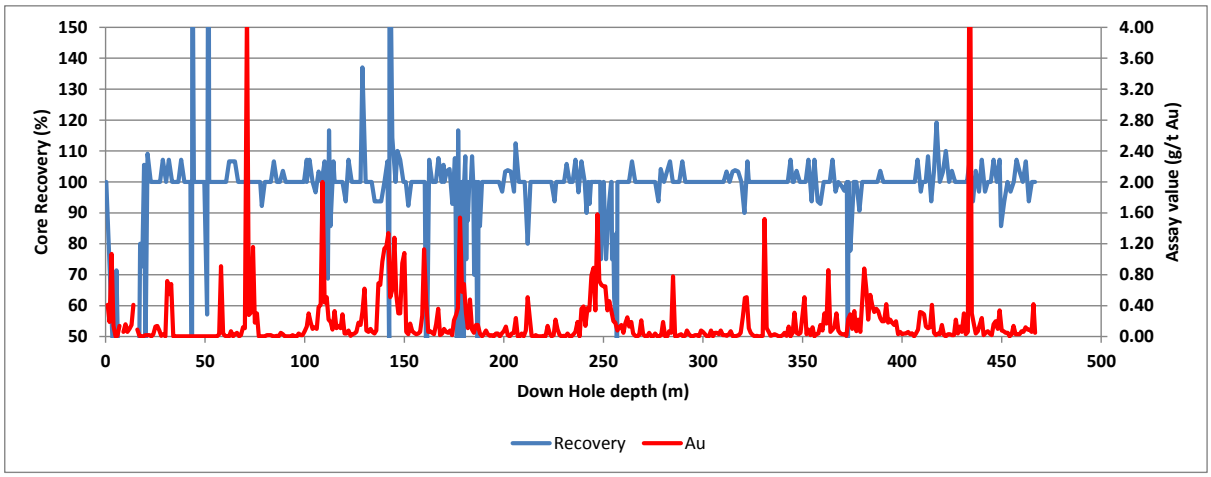


Figure 10.2 *Diamond drill core recovery (blue – left hand axis) versus gold grade (red – right hand axis) for Drill Hole BMVDH014. Diamond core recovery is generally very good. A complex relationship between core recovery and mineralization is present. Zones of reduced core recovery are present between 175 – 186 m and 248 – 258 m which correspond with mineralized intervals. Zone of reduced core recovery between 372 – 375 m is not associated with gold mineralization. Low core recoveries between 0 – 22 m are a result of drilling through previously dumped material. (CGA 2012)*

The presence of historical underground workings has generally not presented significant drilling and sample recovery issues. Underground voids are typically narrow (< 3 m) and their approximate position well understood from historical records. Voids encountered during surface drilling during 2009 to present have generally been successfully bridged without significant drill hole deviation or significantly reduced sample recovery from subsequent intervals. Where underground voids have affected with drilling or sampling quality, drill holes have been abandoned.

10.5 Collar Plan

Drill hole collar locations for all drilling within the 2011 MGP mineral resource estimate are illustrated in Figure 10.3. The database includes assays for drill holes completed up to mid-August 2011.

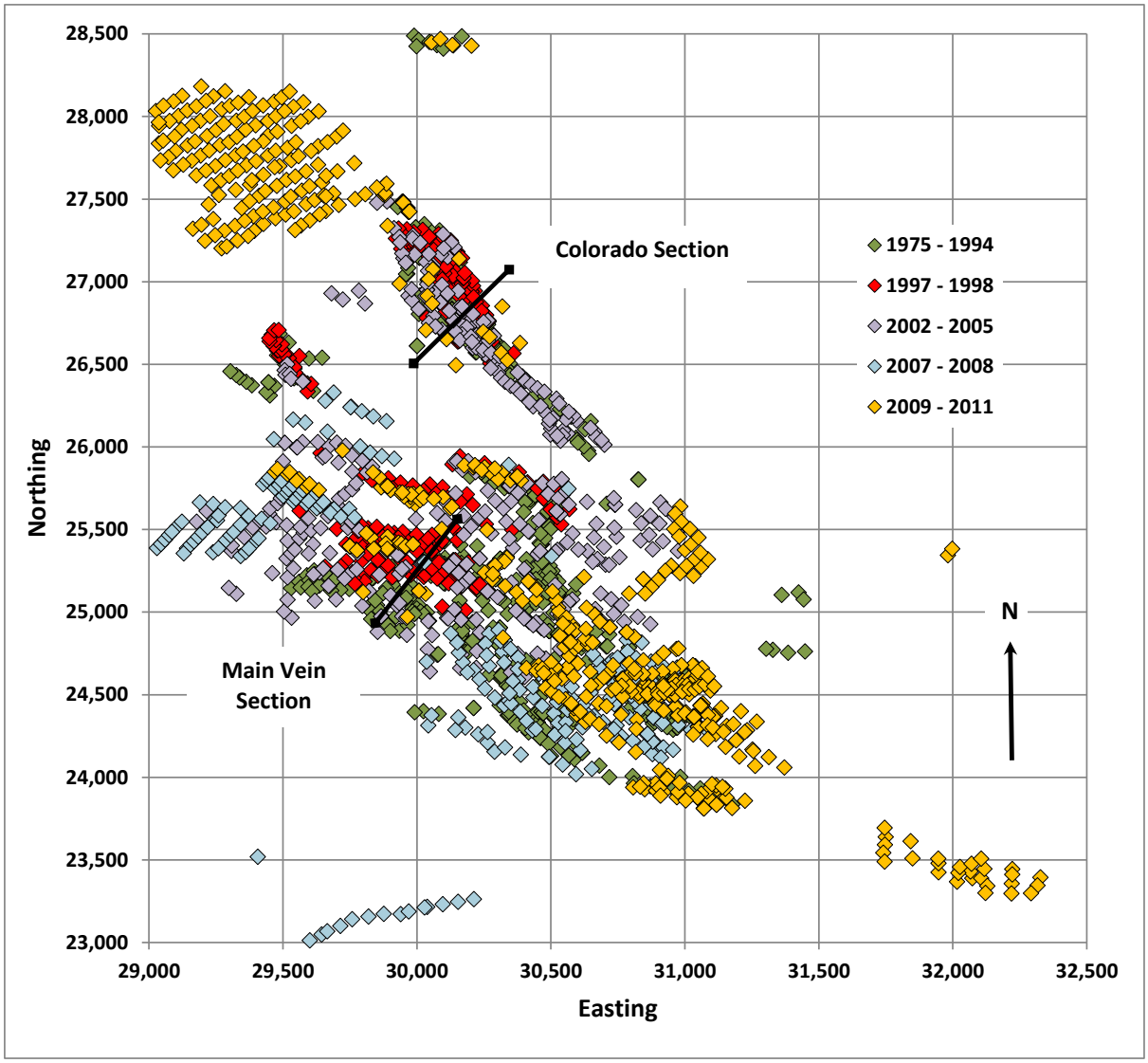


Figure 10.3 *Drill hole collar location (plan view), MGP 2011 mineral resource database. Companies associated with drilling periods = Atlas 1975 – 1994, Base Metals 1997 – 1998, FRC (funded by Thistle Mining) 2002 – 2005, FRC (funded by CGA) 2007 – present. Solid lines illustrate the location of representative sections for Colorado and Main Vein illustrated in Figures 10.4 and 10.5. (CGA 2012)*

10.6 Representative Sections

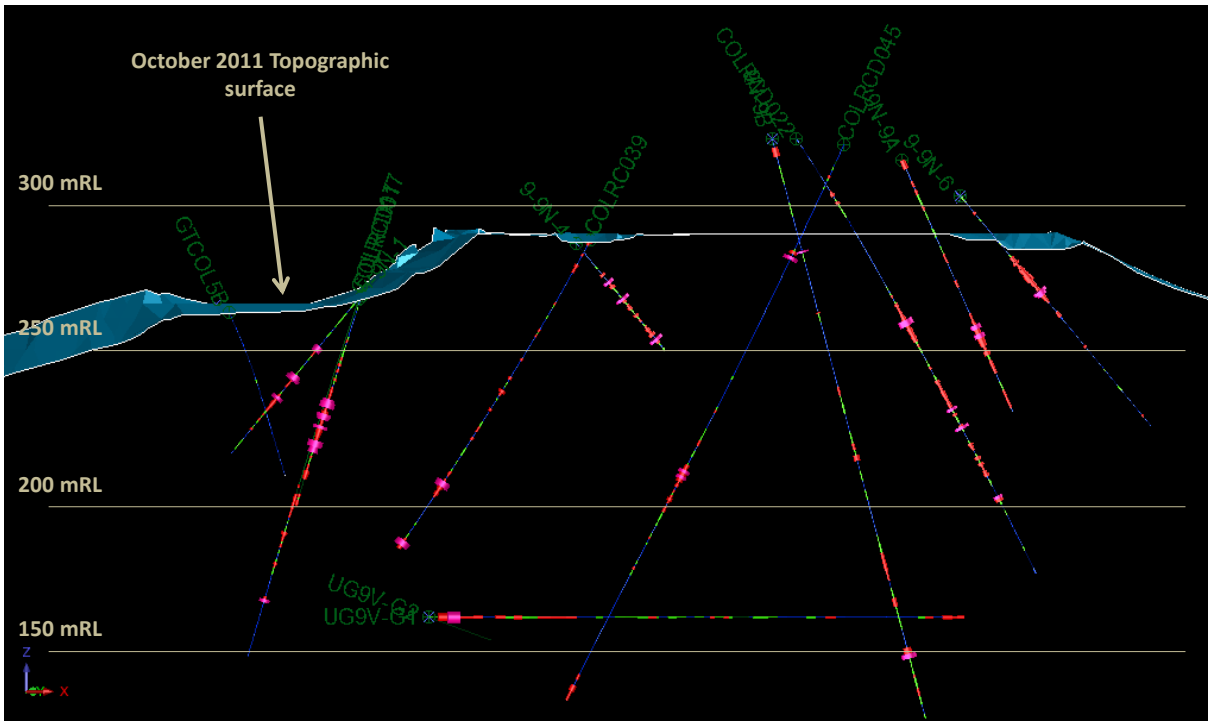


Figure 10.4 *Representative drill section through Colorado deposit. Drill hole colours: blue = < 0.4 g/t Au; green = 0.4 to 0.7 g/t Au; red = 0.7 – 3 g/t Au; magenta > 3 g/t Au. (CGA 2012)*

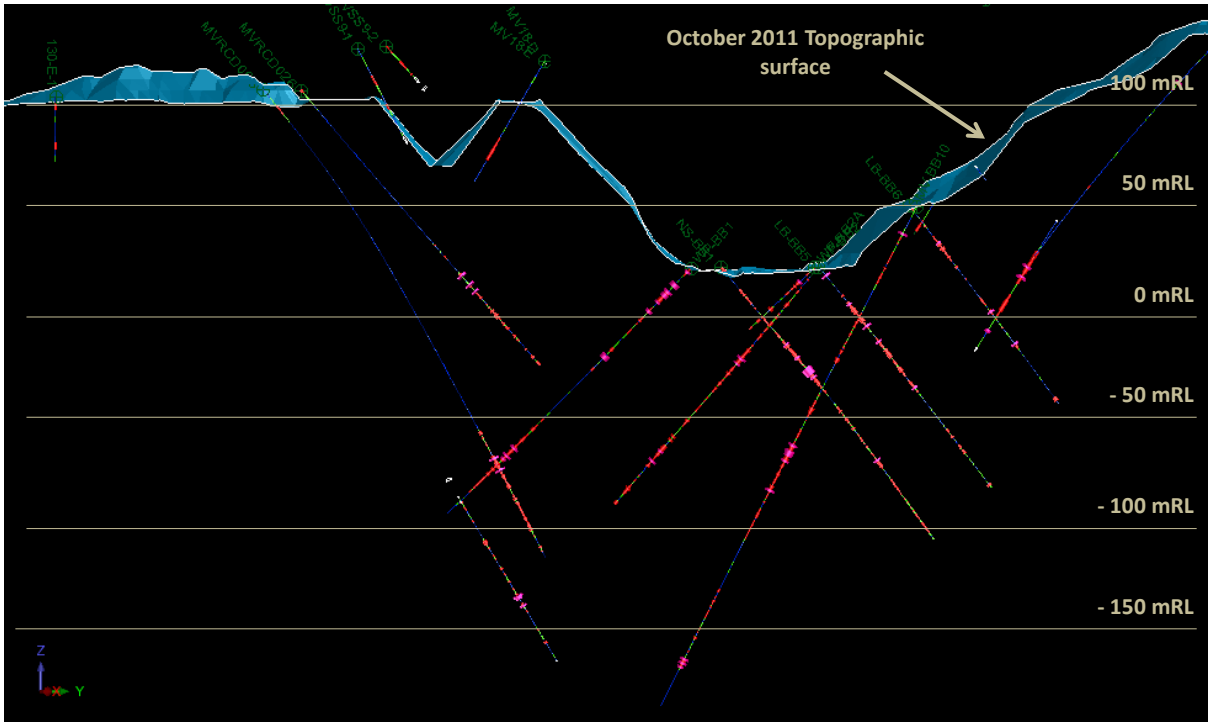


Figure 10.4 *Representative drill section through Main Vein deposit. Drill hole colours: blue = < 0.4 g/t Au; green = 0.4 to 0.7 g/t Au; red = 0.7 – 3 g/t Au; magenta > 3 g/t Au. (CGA 2012)*

10.7 Authors Opinion – Drilling

There has been no independent review of the drill hole surveys and collar locations, although it appears that this work was done to an industry acceptable standard. Further aspects of drilling and sampling are presented in Section 11 and show that no factors were encountered that could materially impact the accuracy and reliability of drilling results.

11 SAMPLE PREPARATION, ANALYSES & SECURITY

11.1 Sample Collection

11.1.1 Historical Sampling Procedures

Diamond

Atlas (1975 – 1994) diamond core was typically sampled in 1.5 m lengths through mineralised zones, although lengths were varied to conform to major geological contacts. Selected intervals of diamond core were identified for assay. The core was not cut – a geological selected 10 cm portion was retained as a record and the entire remaining core crushed to minus ½ inch. The crushed core was spilt and 50% sent to the laboratory for further analysis.

Base Metal (1997 – 1998) diamond core was photographed, logged and sampled at a central core shed. All diamond core was photographed and core recoveries documented prior to cutting and sampling. Broken or very soft core was hand sampled. Once cut, half of the core was retained as a permanent record and half shipped to the laboratory for assay.

Most FRC (2002 – 2005, 2007 – 2008) diamond drill holes were orientated for geotechnical and structural logging using red crayon in a wire line spear. Diamond core was geologically logged and sampled at a central core shed. All diamond core was photographed and core recoveries documented prior to cutting and sampling. Once cut, half of the core was retained as a permanent record and half shipped to the laboratory for assay.

RC

Atlas (1975 – 1994) RC samples were sampled from 1 to 3 m intervals with mixing and splitting of the samples at the drill site using a tarpaulin.

Base Metal (1997 -1998) RC samples were taken at 1 m down hole intervals. The sample was collected, weighed and riffle split. Approximately a 5 kg split was sent for assay with the remaining reject retained for storage. Drill cuttings were logged following mineralogical examination of a small washed sample.

FRC (2002 – 2005, 2007 – 2008) RC samples were collected in 1 m down hole intervals using a cyclone. The entre sample interval was collected in plastic bags, weighed and a 5 kg assay split taken using a Jones riffle splitter. The reject sample was bagged and stored on site. 350 g field samples were taken using a sieve during splitting and washed for geological logging. Chip samples were retained in trays for future reference. On some occasions if drilling was believed to be well into waste, 3 m composite assay samples were made up using the splitter. Individual samples within the composite were retained and only assayed if the composite returned a grade > 0.12 g/t Au. Occasional wet samples were dried in trays before weighing and splitting. Wet samples were encountered in narrow fissures and with drill conditioning often the hole could be dried out. Where it became plain that further drill conditioning could not dry the hole, the remainder was completed using diamond drilling.

11.1.2 Current Sampling Procedures

RC samples are collected in 1 m down hole intervals using a cyclone. The entre sample interval is collected in plastic bags, weighed and a 3 kg assay split taken using a Jones riffle splitter. The reject sample is bagged and stored on site. 350 g representative samples are taken using a sieve during splitting and washed for geological logging (Figure 11.1). Chip samples are retained in trays for future reference. Wet sample intervals are dried before weighing and splitting. RC drill holes that cannot continue to consistently return dry drill cuttings are completed using diamond drilling.

Diamond drill holes are orientated for geotechnical and structural logging using red crayon in a wire line spear. Diamond core is marked into 1 m intervals before being geologically logged, photographed and sampled at a central core shed. The core is marked longitudinally to guide cutting, aligned by matching broken pieces and the portion selected for assay identified. Once cut, half of the core was retained as a permanent record and half shipped to the laboratory for assay.

Retained diamond core and RC chip trays are identified by Drill hole ID and stored at a central warehousing facility (Figure 11.1).

Assay samples are identified by a sequential number and two letter prefix system, with individual sample numbers reserved for blanks and control standards. Batches of assay samples (RC and diamond) are prepared at the core shed facility prior to dispatch to the laboratory by experienced staff (Figure 11.1).

Figure 11.1 *Activities at central core shed facility, MGP. Top right = geological logging of representative RC drill chips; Top left = stacked diamond core; Centre right = stacked RC chip trays; Centre left = diamond core cutting facilities; Bottom centre = Assay sample batch preparation prior to dispatch to laboratory. (MA site visit, 201)*

Grade control samples are collected in 1 m intervals using a cyclone. At the end of each sample interval, cyclone contents are dropped through a rig-mounted Metzke cone splitter to produce 3 kg assay sample. Reject drill cuttings are shoveled back down the open drill hole at the completion of drilling to ensure subsequent blasting operations are not affected. Grade control drill cuttings are not geologically logged.

As with Exploration / Resource drilling, grade control assay samples are identified by a unique sequential number and two letter prefix system, with individual sample numbers reserved for blank and control standards.

11.2 Sample Security

Exploration / resource samples destined for analysis at McPhar Laboratories (McPhar) in Manila are required to be officially inspected prior to dispatch. Inspection requires weighing and sighting of each individual sample by an MGB official. Individual sample shipments from site to Manila are only permitted to a limit of 2,000 kg. Sample dispatches are transported to Manila in sealed containers.

Grade control samples are delivered to the on-site SGS laboratory. The on-site laboratory is located in a high security portion of the MGP, with access of individuals restricted and documented.

11.3 Laboratory Sample Preparation and Analysis

11.3.1 General

Company	Period	Laboratory Location		Laboratory Company
		Sample Prep.	*Assay*	
Atlas	1975 - 1978	Cebu	Cebu	Atlas
Atlas	1980 – 1994	Masbate[1]	Masbate[1]	Atlas
Base Metals	1997 – 1998	Masbate[1]	Masbate[1]	Base Metals
FRC	2002 – 2004	Manila	Manila	McPhar
FRC	2005	Masbate[2]	Manila	McPhar
FRC	2007 – 2008	Manila	Manila	McPhar
FRC	2009 – 2011	Manila	Manila	McPhar
FRC	2011 - present	Masbate[3]	Tianjin, China	SGS
FRC (GC + Mill)	2009 - present	Masbate[4]	Masbate[4]	SGS

Table 11.1 *Summary of laboratory location and companies by period, MGP samples. Masbate[1] = Atlas laboratory located at the Masbate mine site; Masbate[2] = former Atlas mine site laboratory refurbished with new sample prep. equipment; Masbate[3] = shipping container mobile sample prep. unit (MSPU) outfitted with new sample preparation equipment; Masbate[4] = new mine site laboratory facility constructed during 2009.*

A number of laboratories have produced analyses of samples from the MGP (Table 11.1). Sample preparation protocols and assay methods are briefly summarised below:

11.3.2 Atlas

Details of samples collected from drilling conducted by Atlas before 1980 were analysed at the Atlas operated laboratory at Cebu.

During the period 1980 to 1994, Atlas drill samples were analysed at an Atlas operated laboratory locate at the Masbate mine site. All samples were prepared by drying followed by crushing to < 6 mm, split to 1 kg, gyratory crushed, split to 500 g and pulverised in a ring milled for 3 minutes.

The Atlas operated Masbate laboratory processed many run-of-mine samples where the assay method for Au was by AAS, with 1 in 5 (or 10) samples re-assayed by fire assay. From discussions with former Atlas staff, it is understood that exploration samples were all directed to be assayed by fire assay. This is contrary to past documentation (Howe 1996, Base Metals 1997) that indicated all geological assaying was conducted by AAS.

11.3.3 Base Metals

Base Metals assayed all samples at the former Atlas mine site laboratory. Sample preparation protocol included drying followed by jaw crushing to < 6 mm, Bico Badger crushing to < 2mm, split to 500 g using a Jones riffle splitter and pulverised by ring mill to < 200 mesh (75 micron).

All gold determinations were completed by fire assay. Independent re-analysis and statistical review by BDA during 1997 identified a low bias in Base Metal assay values. The cause of the bias was eventually identified as being caused by the use of home-made cupels during the assay process and commercial cupels were utilised in 1998 assays.

11.3.4 McPhar

Exploration \ resource samples from MGP have been analysed by McPhar Geoservices Philippines Incorporated (McPhar) in Manila since 2002. McPhar is independent of both FRC and CGA. McPhar is owned by Intertek Overseas Holdings Limited and is ISO 9001:2000 accredited with ISO 17025 accreditation in progress.

FRC (funded by Thistle Mining) sent all samples for analysis to McPhar's Manila laboratory during 2002 – 2004. McPhar's sample preparation included drying followed by jaw crushing of diamond core samples. The entire sample was pulverised in a ring mill to 90% passing – 200 mesh (75 micron) before mixing and serial splitting with a riffle splitter to a 250 g pulp sample. 5% of samples were rechecked for pulverised size and a quartz flush was used to clean the pulveriser after each sample.

During late 2004, FRC decided to refurbish part of the Atlas mine site laboratory and complete sample preparation on site to reduce costs and improve assay turnaround time. Atlas era sample preparation was replaced and McPhar commissioned to manage and supervise the facility using the sample preparation protocol utilised at the McPhar Manila laboratory. Pulp samples were shipped to Manila for analysis.

The refurbished former Atlas mine site laboratory was decommissioned before the commencement of FRC drilling commencing in 2007. From 2007 onwards, assay samples have been sent to Manila for sample pre and assay. The most recent McPhar sample preparation protocol is presented in Figure 11.2.

All samples during the 2002 – 2005 period were analysed using fire assay of a 50 g charge and an AAS finish. The stated detection limit was 0.01 g/t. The most recent McPhar assay preparation protocol is presented in Figure 11.3.

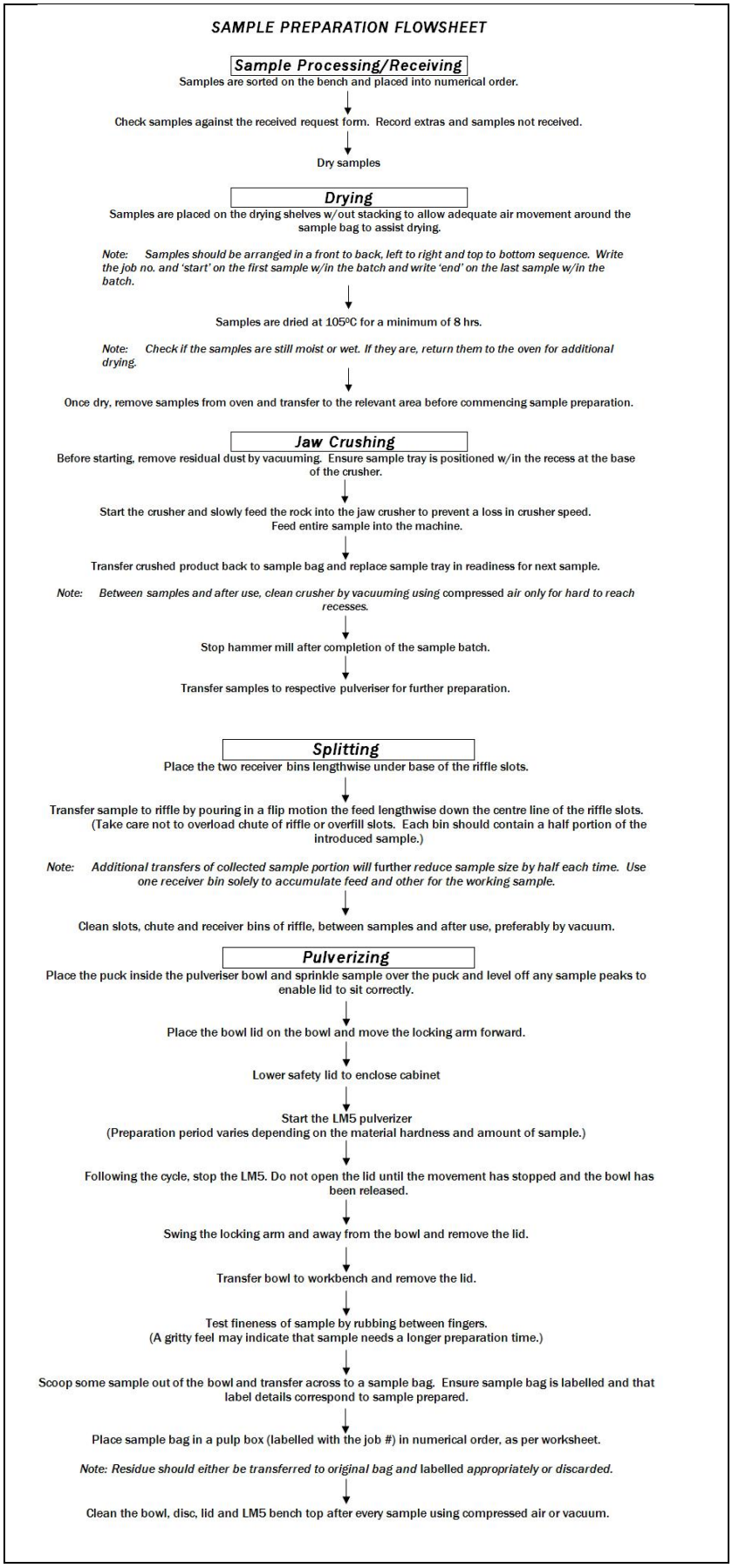

SAMPLE PREPARATION FLOWSHEET
Sample Processing/Receiving
Samples are sorted on the bench and placed into numerical order.
Check samples against the received request form. Record extras and samples not received.
Dry samples
Drying
Samples are placed on the drying shelves w/out stacking to allow adequate air movement around the sample bag to assist drying.
Note: Samples should be arranged in a front to back, left to right and top to bottom sequence. Write the job no. and 'start' on the first sample w/in the batch and write 'end' on the last sample w/in the batch.
Samples are dried at 105ºC for a minimum of 8 hrs.
Note: Check if the samples are still moist or wet. If they are, return them to the oven for additional drying.
Once dry, remove samples from oven and transfer to the relevant area before commencing sample preparation.
Jaw Crushing
Before starting, remove residual dust by vacuuming. Ensure sample tray is positioned w/in the recess at the base of the crusher.
Start the crusher and slowly feed the rock into the jaw crusher to prevent a loss in crusher speed. Feed entire sample into the machine.
Transfer crushed product back to sample bag and replace sample tray in readiness for next sample.
Note: Between samples and after use, clean crusher by vacuuming using compressed air only for hard to reach recesses.
Stop hammer mill after completion of the sample batch.
Transfer samples to respective pulveriser for further preparation.
Splitting
Place the two receiver bins lengthwise under base of the riffle slots.
Transfer sample to riffle by pouring in a flip motion the feed lengthwise down the centre line of the riffle slots. (Take care not to overload chute of riffle or overfill slots. Each bin should contain a half portion of the introduced sample.)
Note: Additional transfers of collected sample portion will further reduce sample size by half each time. Use one receiver bin solely to accumulate feed and other for the working sample.
Clean slots, chute and receiver bins of riffle, between samples and after use, preferably by vacuum.
Pulverizing
Place the puck inside the pulveriser bowl and sprinkle sample over the puck and level off any sample peaks to enable lid to sit correctly.
Place the bowl lid on the bowl and move the locking arm forward.
Lower safety lid to enclose cabinet
Start the LM5 pulverizer
(Preparation period varies depending on the material hardness and amount of sample.)
Following the cycle, stop the LM5. Do not open the lid until the movement has stopped and the bowl has been released.
Swing the locking arm and away from the bowl and remove the lid.
Transfer bowl to workbench and remove the lid.
Test fineness of sample by rubbing between fingers.
(A gritty feel may indicate that sample needs a longer preparation time.)
Scoop some sample out of the bowl and transfer across to a sample bag. Ensure sample bag is labelled and that label details correspond to sample prepared.
Place sample bag in a pulp box (labelled with the job #) in numerical order, as per worksheet.
Note: Residue should either be transferred to original bag and labelled appropriately or discarded.
Clean the bowl, disc, lid and LM5 bench top after every sample using compressed air or vacuum.

Figure 11.2 *Preparation protocol for MGP samples, McPhar laboratories.*

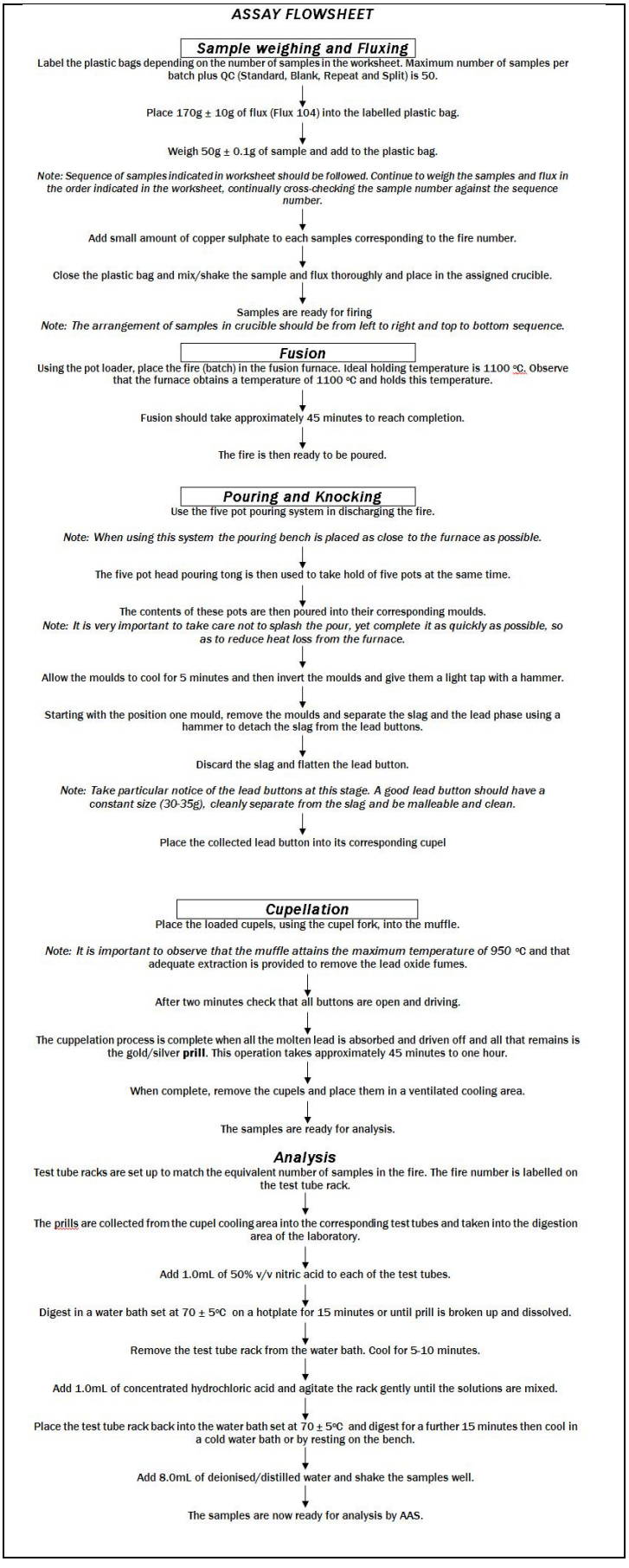

ASSAY FLOWSHEET
Sample weighing and Fluxing
Label the plastic bags depending on the number of samples in the worksheet. Maximum number of samples per batch plus QC (Standard, Blank, Repeat and Split) is 50.
Place 170g ± 10g of flux (Flux 104) into the labelled plastic bag.
Weigh 50g ± 0.1g of sample and add to the plastic bag.
Note: Sequence of samples indicated in worksheet should be followed. Continue to weigh the samples and flux in the order indicated in the worksheet, continually cross-checking the sample number against the sequence number.
Add small amount of copper sulphate to each samples corresponding to the fire number.
Close the plastic bag and mix/shake the sample and flux thoroughly and place in the assigned crucible.
Samples are ready for firing
Note: The arrangement of samples in crucible should be from left to right and top to bottom sequence.
Fusion
Using the pot loader, place the fire (batch) in the fusion furnace. Ideal holding temperature is 1100 °C. Observe that the furnace obtains a temperature of 1100 °C and holds this temperature.
Fusion should take approximately 45 minutes to reach completion.
The fire is then ready to be poured.
Pouring and Knocking
Use the five pot pouring system in discharging the fire.
Note: When using this system the pouring bench is placed as close to the furnace as possible.
The five pot head pouring tong is then used to take hold of five pots at the same time.
The contents of these pots are then poured into their corresponding moulds.
Note: It is very important to take care not to splash the pour, yet complete it as quickly as possible, so as to reduce heat loss from the furnace.
Allow the moulds to cool for 5 minutes and then invert the moulds and give them a light tap with a hammer.
Starting with the position one mould, remove the moulds and separate the slag and the lead phase using a hammer to detach the slag from the lead buttons.
Discard the slag and flatten the lead button.
Note: Take particular notice of the lead buttons at this stage. A good lead button should have a constant size (30-35g), cleanly separate from the slag and be malleable and clean.
Place the collected lead button into its corresponding cupel
Cupellation
Place the loaded cupels, using the cupel fork, into the muffle.
Note: It is important to observe that the muffle attains the maximum temperature of 950 °C and that adequate extraction is provided to remove the lead oxide fumes.
After two minutes check that all buttons are open and driving.
The cuppelation process is complete when all the molten lead is absorbed and driven off and all that remains is the gold/silver prill. This operation takes approximately 45 minutes to one hour.
When complete, remove the cupels and place them in a ventilated cooling area.
The samples are ready for analysis.
Analysis
Test tube racks are set up to match the equivalent number of samples in the fire. The fire number is labelled on the test tube rack.
The prills are collected from the cupel cooling area into the corresponding test tubes and taken into the digestion area of the laboratory.
Add 1.0mL of 50% v/v nitric acid to each of the test tubes.
Digest in a water bath set at 70 ± 5°C on a hotplate for 15 minutes or until prill is broken up and dissolved.
Remove the test tube rack from the water bath. Cool for 5-10 minutes.
Add 1.0mL of concentrated hydrochloric acid and agitate the rack gently until the solutions are mixed.
Place the test tube rack back into the water bath set at 70 ± 5°C and digest for a further 15 minutes then cool in a cold water bath or by resting on the bench.
Add 8.0mL of deionised/distilled water and shake the samples well.
The samples are now ready for analysis by AAS.

Figure 11.3 *Assay protocol for MGP samples, McPhar laboratories.*

11.3.5 SGS

FRC (funded by CGA) in 2009 entered into a contract with SGS Philippines Incorporated (SGS) to provide analytical services to support grade control and ore processing operations at MGP from purposed built new laboratory building located adjacent to the Masbate Process plant. SGS Philippines is part of the SGS Global group and is independent of both FRC and CGA.

In 2011, FRC also entered into agreement with SGS to provide analysis of exploration \ resource samples. Samples are prepared at on site mobile sample preparation unit (MSPU) which is independent of the SGS operated mine laboratory building. Pulps are then shipped for analysis at the SGS Taiwan Limited (SGS Taiwan) facility in Tianjin, China. The Tianjin facility has ISO 9001 and ISO/IEC 17025 accreditation. SGS commenced processing of MGP exploration \ resource samples in June 2011.

Sample preparation includes drying followed by jaw crushing to 75% passing 2 mm, split to < 1.5 kg, and pulverisation of 1.5 kg split to 85% passing 75 micron. Every 50th sample is screened to confirm % passing 2 mm and 75 micron. The crusher and pulveriser are cleaned with barren material at the start of every batch and after every 50th sample. The MPSU facility is staffed, managed and supervised by SGS personnel.

Determination of Au content is by fire assay of a 50 g charge with an AAS finish. The stated detection limit is 0.01 g/t.

11.4 QAQC

Quality assurance (“QA”) and quality control (“QC”) measures are routinely employed to monitor the accuracy and reproducibility of laboratory preparation and assay results.

11.4.1 Pre 2002 QAQC

QAQC measures implemented within the Atlas operated mine site laboratory included blank samples, laboratory standards analysed at the rate of approximately 1 in 10 to 15 sample and participation in systematic round-robin checks exchanged with Philex, Banguet and the Atlas-operated Cebu laboratory. Comparisons were reported as generally good with no indication of systematic high or low bias.

Base Metal routine QAQC measures included resubmission of samples as an internal check at a rate of 1 in 15, blank and standard samples inserted at a rate of 1 in 30 samples and re-analysis of any sample in excess of 10 g/t Au. In addition, 15 samples were selected per month were submitted to McPhar in Manila and Chemex in Canada.

As mentioned in section 11.3.3, independent assay of check samples and statistical analysis of inter laboratory results indicate 1997 Base Metal assays have a low bias as a result of the use of home-made cupels during the assay process. Approximately 6,900 samples from the 1997 drilling campaign were re-analysed using commercial cupels during 1998. Re-analysed samples concentrated on mineralised interval and were systematically higher than the original 1997 result. Independent review (Minproc 1998, BDA 1998) established that although original assay values are uniformly low, the magnitude of the bias is not constant across all grade ranges or sample types.

Within the 1998 and 2002 resource estimates, Base Metal assay data was adjusted upwards by 16% following a recommendation by Minproc. Following detailed analysis, IMC and MA consultants disagreed with this approach and MGP mineral resource estimates since 2004 have been calculated using unadjusted assay data. Unadjusted assay data has been used in calculation of the 2011 MGP mineral resource estimate.

In 2011 it was recognised by FRC staff that assay data generated by the Masbate mine site laboratory (i.e. all assay data prior to 2002) is characterised by a very few results below 0.25 g/t Au and suggest the effective lower limit of detection was in the range of 0.2 to 0.3 g/t. Historical QAQC measures and inter laboratory checks indicate assay values above 0.3 g/t are accurate and (with the exception of 1997 Base Metal assays) unbiased. A potential result of the elevated 'effective" lower limit of detection may be an over estimation of the amount of low grade or sub economic mineralised material within the "Halo" domain of the MA's 2008 MGP mineral resource estimate. As a conservative measure, no pre 2000 assay data has been used to estimate the grade of Halo material in the 2011 MGP mineral resource estimate.

11.4.2 2002 – 2008 QAQC

Routine QAQC measures at McPhar laboratories include duplicate analysis of 1 in 10 samples, analysis of a commercial standard at the rate of 1 in 30 samples, analysis of certified blank material at a rate of 1 in 100 samples and regular participation in inter-laboratory round robin analysis. No obvious laboratory bias or error is apparent in McPhar results.

In 2005, IMC due diligence sampling re-assayed selected samples at both McPhar and an independent laboratory in Western Australia (Ultra Trace Pty Ltd). Results demonstrated there was no evidence of assay bias in McPhar results and that the introduction of sample preparation at the Masbate mine-site in 2005 had no material effect on the quality of McPhar assay data.

11.4.3 2009 – 2011 QAQC

FRC requested Rock Solid Data Consultancy Pty Ltd ("Rock Solid Data") to compile a QC Report based on resource holes completed since 2009 and holes with prefix RCGC in preparation for resource modelling. The QC report is contained in Appendix 1. The following descriptions are extracted from the Executive Summary of the QC Report.

The QC report was based on the quality control data associated with the 622 RC and diamond resource drill holes completed between Apr 2010 and Sep 2011 and 5168 RCGC drill holes completed between Jun 2008 and Apr 2010. FRC collected 60,000 resource RC samples, 13,000 resource diamond samples and over 130,000 RCGC samples from these drill holes. Available quality control data associated with drill samples include field duplicates, laboratory duplicates and analytical repeats along with certified reference material and blanks. The samples were sent to McPhar Laboratories Manila Philippines (McPhar) and SGS Masbate onsite Laboratory (SGS) for analysis by Fire Assay AAS method.

Rock Solid Data consider that overall the results are good and provide a reasonable level of confidence in the data. The key issue identified by Rock Solid Data is a low bias in the SGS gold results for certified standards when compared with the expected value; in all cases the low bias is not present in McPhar gold results for the same standards. The bias is present in a few Geostats

standards as well as two of the four recently adopted Rocklabs standards. The main concern with a low bias is the affect it would have on a resource model. The other key issue is Standard ID mis-identification which is occurring in even the most recent Rocklabs standards; these significant outliers are impacting on the statistics making it difficult to distinguish between true laboratory bias and bias caused by the effect of outliers.

Rock Solid Data recommended continuous quality control monitoring on a daily and monthly basis; any issues identified need to be investigated and addressed, especially issues relating to biases. Should a laboratory bias be detected the responsible geologist must immediately follow-up with the laboratory to determine the cause and solution. Standard outliers should be investigated to firstly determine if Standard ID mislabelling is the cause, and the database corrected if necessary; outliers can disguise true laboratory biases in statistics, so it is critical to minimise these obvious mistakes through ongoing monitoring and management.

The results of Rock Solid Data's statistical analysis are summarised in the following graphs in Appendix 1:

- Standard Control Plots which show the assay results of a particular reference standard over time. The results can be compared to the expected value and 2x and 3x the expected standard deviations, providing a good indication of both precision and accuracy over time.
- Correlation Plots which are simple plots of the value of the original assay against the repeat assay. The plot allows an overall visualisation of precision and bias over selected grade ranges. The plot is present in normal scale and log scale.
- Particle Size Analysis graphs which show a month-by-month trend of the particle size analysis results of the prepared sample. The plot is a good indicator of the performance of the laboratory's sample preparation techniques.

11.5 Authors Opinion – Current Practice

There has been no independent review of the drill hole sampling, geological logging and geological interpretations done by MA. Although it appears that this work was done to an industry acceptable standard, there is always a risk involved with geological interpretations and grade continuity.

Generally, the results of the QA/QC program implemented by FRC are considered good and provide a reasonable level of confidence in the data. It is MA's opinion that the sample preparation, security and analytical procedures were adequate and follow accepted industry standards.

12 DATA VERIFICATION

12.1 Data Verification Processes

12.1.1 Site Visits

Mr Vigar first visited the site in 1996 and 1997 and again on 5th to the 8th of August 2011. MA conducted the previous mineral resource estimates (and site visits) described in the 2006 MA NI43-101 and the 2008 MA NI43-101 reports.

Mr Steven Potter, an employee of MA, visited the property from 29 Jan to 4 Feb, 2005 (Potter, 2005a), 1-6 April 2005 (Potter, 2005b) and again in 1-14 July 2005. Mr I Taylor, an employee of MA visited the project in March 2011 after mining operations had commenced.

12.1.2 Independent Sampling

No independent samples were collected for this 2011 report.

12.1.3 Database Verification

Mr Vigar has reviewed all new data since the 2008 MA NI43-101 report relevant to the areas of the new mineral resource estimation described in this report. Mr Vigar previously verified the data contained in the 2008 NI43-101 Technical Report.

12.2 Limitations on Verification

A certain amount of historical data has been included in the estimates, and records for some of these data do not have supporting QA/QC information. This is discussed at length elsewhere in this report.

The project is currently in production and results are generally in line with expected outcomes.

12.3 Verification Opinion

The exploration program and resource drilling conducted by FRC's on the MGP is being carried out at industry standards. The QA/QC procedures adopted for the submission of drill samples are industry standard.

Based on the data verification performed, it is MA's opinion that the data reviewed is adequate for the purposes used in this technical report.

13 MINERAL PROCESSING & METALLURGICAL TESTING

13.1 General

Metallurgical data for the MGP has been determined from a number of diagnostic test programs in addition to the processing ore during historic and ongoing operations (Atlas 1980 to 1994 and PGPRC 2009 to present).

There is limited metallurgical data available from Atlas processing operations. Atlas processed a significant volume of material over a long period of time (17.4 Mt processed over 14 years for in excess of 1 Moz Au production) and proved that gold could be economically recovered from a number of ore types at the MGP using a cyanide leaching process.

Comprehensive metallurgical test programs were conducted as part of Feasibility Studies competed in 1998 and 2006. Testing involved comminution and gold recovery characterisation sufficient to provide process flow sheet criteria required for detailed plant design and were conducted according to industry best practice. General conclusions drawn for the test work performed on Masbate ore samples were:

- Investigations indicate that the metallurgical response of the various vein systems at Masbate is complex and variable. MGP ores are, however, amenable to gold extraction by a conventional cyanidation process. From the recovery response obtained for the cyanidation test work, the material can be categorised as either free-milling or mildly refractory;
- For the leach residue grades and sulphur grades obtained, whole ore cyanidation offered the optimum economic outcome for the project;
- The grinding work indices, rheology, settling characteristics and metallurgical response were found to vary with the level of oxidation;
- The gold grade present in the leach residue is dependent on the level of ore oxidation. Residue grades generally increase as the level of oxidation decreases;
- The residue grades obtained for the material from the transition and primary zones were related to the feed sulphur level. Residue grades generally increased as the level of sulphur increased;
- A range of abrasion indices were obtained for the samples tested. The results indicate that the Masbate ore types are expected to yield a moderate to high consumption of grinding media, liners and crushing consumables; and
- Gravity test work resulted in very low gold recoveries to gravity concentrates. A gravity circuit was not recommended for inclusion in process plant design.

13.2 Comminution

In 2005 several samples of diamond core selected from drill hole intervals located throughout the Masbate mineral resource were tested to define the comminution properties. Samples were tested at specialised facilities operated by Metcon Laboratories (Sydney, Australia) and AMMTEC Limited (Perth, Australia).

Ore from Masbate can generally be classified as Medium Hard. Results used for the design the grinding circuit of the current Masbate Process plant are presented in Table 13.1.

Sample ID	A_i	CW_i (kWh/t)	RW_i (kWh/t)	BW_i (kWh/t)	A	b	A*b	ta	SG
OXIDE – Col/GV	0.293	10.0	11.7	11.1	79.6	0.99	78.8	0.79	2.4
TRANSITION ORE									
Col/GV 1	0.137	12.0	15.2	14.7	75.7	0.86	65.1	0.65	2.47
Col/GV 1	0.705	12.0	21.2	20.3	100.0	0.42	42.0	0.42	2.49
PRIMARY ORE									
Colorado 98	0.600	15.0	16.4	16.7	70.3	0.59	41.5	0.51	2.65
Basalt 98	0.430	15.0	19.3	19.7	73.0	0.48	35.0	0.41	2.65
Main Vein 98	0.540	15.0	14.3	15.7	67.2	0.8	53.7	0.44	2.65
Main Vein Primary 1	0.210	15.0	20.6	19.5	82.4	0.58	47.8	048	2.58
Main Vein Primary 2	0.625	15.0	18.5	18.6	85.7	0.63	54.0	0.54	2.59
Main Vein Primary 3	0.344	15.0	16.7	17.3	81.4	0.73	59.4	0.59	2.58
Main Vein Primary 4	0.549	15.0	20.0	19.0	94.5	0.47	44.4	0.44	2.60
Main Vein Primary 5	0.347	15.0	16.7	15.6	77.9	0.87	67.8	0.67	2.63
HMB – Primary 1	0.328	15.0	19.5	19.7	83.2	0.54	44.9	0.44	2.61
HMB – Primary 2	0.390	15.0	18.4	16.5	93.0	0.50	46.5	0.47	2.57
HMB – Primary 3	0.355	15.0	20.0	23.6	89.2	0.47	41.9	0.42	2.63

Table 13.1 *Summary of Comminution results, AMMTEC Laboratories 2005. All parameters experimentally measured except Crushing Work Index (CW_i) – values for this parameter was are assumed based on ore characteristics obtained from RW_i and BW_i. Col = Colorado, GV = Grand View, HMB = Holy Moses Basalt.*

13.3 Gold Recovery

Gold recovery testing was completed on drill hole samples located throughout the 2006 MGP mineral resource for the 2006 Feasibility Study. Since the commencement of the processing operation 2009, a significant amount of in house metallurgical testing has been conducted to maximise gold extraction from the current circuit.

13.3.1 2006 Feasibility

Recovery characteristics for MGP ore were completed under supervision by Ausenco at Metcon Laboratories (Sydney, Australia) on a comprehensive set of samples representing individual ore zones, composite ore zones and mill feed (i.e. blended samples) with goal of establishing the most appropriate leach conditions for treatment of MGP ore. Scoping studies established that degree of oxidation and head grade had the most significant effect on gold recovery. Experimental test work investigated recovery variation due to grind size, leach time, and cyanide concentrate, as well as documenting leach kinetics, cyanide and lime consumption, Ag recovery, slurry rheology, carbon adsorption and cyanide detoxification.

Of the parameters tested, the most significant recovery relationship was reacted to grind size, with a general reduction in residual grade (hence increasing gold recovery) with reducing grind size recorded from all composite samples (Figure 13.1).

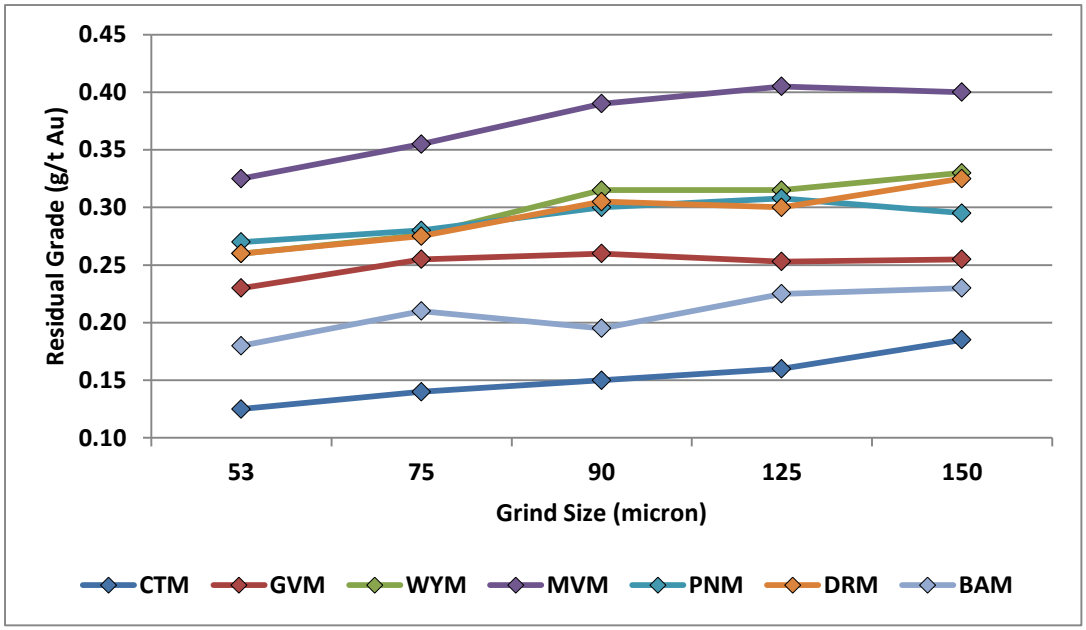


Figure 13.1 *Experimental results of residual grade (left hand axis = g/t Au) versus P_{80} grind size (in microns) for 6 composite ore samples from MGP. Individual test conditions standardized to leach time of 24 hours with initial cyanide concentration of 1,000 ppm. CTM = Colorado composite ; GVM = Grandview composite; WYM = Wyoming composite; MVM = Main Vein composite; PNM = Panique composite; DRM = Doris composite; BAM = Basalt composite. Data sourced from Ausenco (2006). (CGA 2012)*

Leach testing of composite ore zone samples showed that between 50 – 70% of gold was extracted in the first 8 hours and that residual gold is very slowly leached after 24 hours (Figure 13.2).

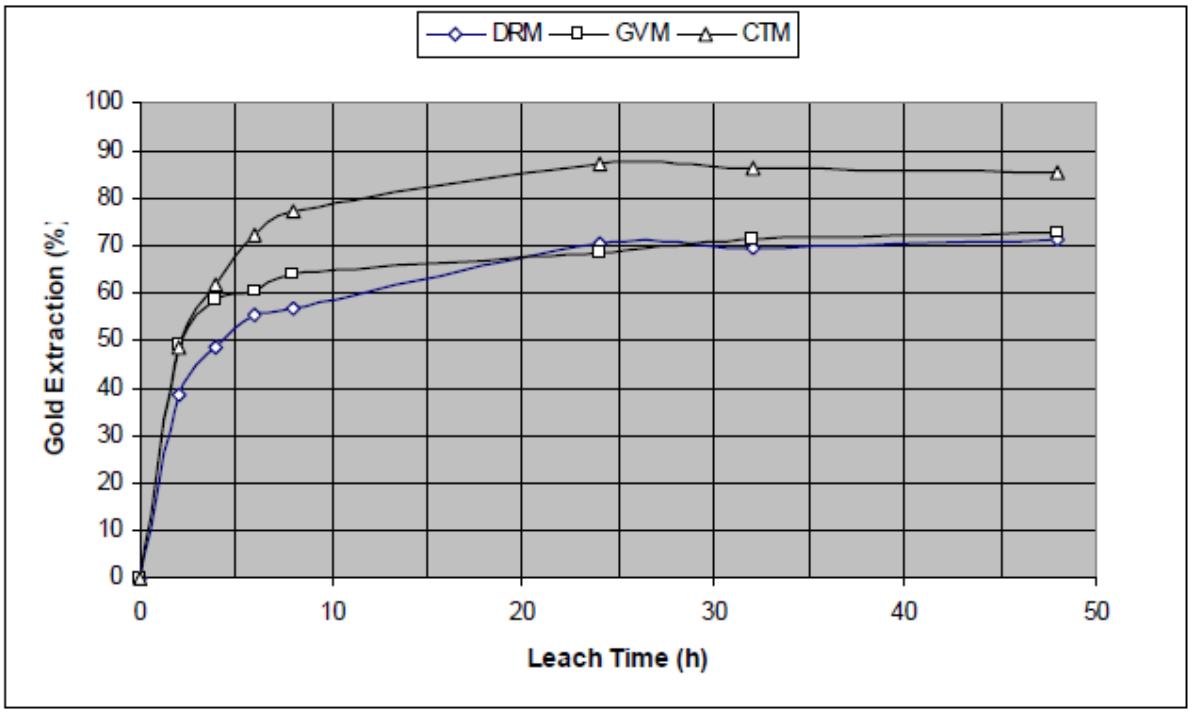


Figure 13.2 *Gold dissolution for composite ore zone samples, MGP. Tests conducted for 48 hours, at 50% solids and 0.1% NaCN. CTM = Colorado composite ; GVM = Grandview composite; DRM = Doris composite. Sourced from Ausenco (2006).*

On the basis of an economic analysis using a gold price of 390 US$ per ounce, Ausenco selected a process flow of P_{80} of 150 micron and a leach time of 24 hours.

The average results for testing of samples from individual ore zone samples under this process flow are presented in Table 13.2.

Material	No. Samples	Head Grade	Residual Grade	Recovery	Reagent Consumption (kg/t)	
		g/t Au	*g/t Au*	*%*	*NaCN*	*Lime*
Oxide	13	1.24	0.137	88.9	0.52	3.50
Transition	16	1.20	0.217	81.8	0.61	3.12
Primary	18	1.20	0.361	70.0	0.55	0.85

Table 13.2 *Summary of testing of individual ore zone samples, MGP. Testing conducted at grind size of P_{80} of 150 micron and leach time of 24 hours. Sourced from Ausenco (2006).*

CGA utilised much of the Ausenco metallurgy data in design and construction of the Masbate Process plant. Purchase of second hand grinding mills (SAG and 2 x Ball Mills) and a rising gold price however prompted a decision to target a finer grind size (P_{80} of 106 micron).

13.3.2 Lead Nitrate

In house laboratory testing indicated that the use of lead nitrate as a reaction catalyst had a positive effect on gold recovery from MGP ore. Although generally only resulting in a modest recovery increase over a 24 hour leach time (Table 13.3), the use of lead nitrate significantly increased the amount of gold dissolution in the first 8 to 10 hours of leaching. Trials of lead nitrate in the process plant commenced in early 2010 and continue to the present.

Sample	Material	Head Grade (g/t Au)	Residual Grade (g/t Au)		Recovery (%)	
			CIL	*CIL + $PbNO_3$*	*CIL*	*CIL + $PbNO_3$*
Libra	Primary	1.59	0.42	0.34	75.95	76.51
Holy Moses	Primary	2.07	0.57	0.45	73.05	77.74
Libra	Transition	2.14	0.47	0.22	79.18	89.16
Holy Moses	Transition	0.91	0.14	0.09	86.74	90.75
Montana	Oxide	1.16	0.15	0.12	87.07	89.66

Table 13.3 *Effect of lead nitrate on selected ore zone samples, MGP. Tests conducted on material with grind size P_{80} passing 106 micron, 24 hour leach time and 0.55 kg/t NaCN. 2009 to 2011 in house test work, PGPRC.*

As throughput rates through the Masbate Process plant have increased above name plate (4 Mtpa), residence time in the CIL circuit has reduced from 24 hour (refer to section 17 for further details). The use of lead nitrate has allowed the reduction in residence time to occur without penalty to gold recovery.

13.3.3 Production Recovery

Production recovery rates are discussed Section 17 and illustrated by month in Figure 17.3. As demonstrated by production performance, gold can and will continue to be economically recovered from MGP ore in the current Masbate Process plant. No major changes in metallurgy with increasing depth are suggested from the available test work.

Based on metallurgical test work and production history, recovery values used for forward planning are presented in Table 13.4

Head Grade	Oxide	Transition	Primary
<1.0 g/t	85%	82%	74%
>1.0 g/t	93%	86%	82%

Table 13.4 *Predicted gold recovery by material type used for forward planning purposes, MGP.*

14 MINERAL RESOURCE ESTIMATES

14.1 Approach

The current 2011 resource estimate involved the following activities:

- Database compilation and review (including QAQC).
- Generation of topographic files showing pre-mine, mined out, back-filled and dumps.
- Comparisons of drill data statistics with grade control and production data over the 6 month period to September 2011.
- New wireframes for the vein and halo domains.
- Use of the grade control drilling, where available.
- Tagging of all relevant drill intercepts by domain to the drill database.
- Constructing a new block model covering all domains.
- Grade estimation using ordinary kriging with block size varied by block size.
- Assignment of bulk density by domain and weathering.
- Validation.
- Assignment of resource categories based on estimation parameters.
- Producing a revised resource estimation reporting.

14.2 Supplied Data

MA was supplied with two of CGA's current drill databases (ds_exgt.ddb and gch.ddb). The databases structures are displayed in Table 14.1.

Table Name	Description
Alteration	Zones and associated minerals of alteration
Assays	Drill hole assay results
Collar	Location of hole id and collar coordinates
Lithology	Drill hole lithology
Ma_code	Mining Associates corresponding codes
Minerals	Mineralisation type and location
Oxidation	Oxidation type and location
Styles	Styles types as used in modeling
Surveys	Down hole drill holes survey data
Veins	Vein type and location
Veinsview	Percentage of vein type and vein style
Styles	Styles types as used in modeling
Assays	Drill hole assay results
Collar	Location of hole id and collar coordinates
Surveys	Down hole drill holes survey data
Veinsview	Percentage of vein type and vein style
Code_gc	Mineralised zone codes and location

Table 14.1 *Database structure, 2011 MGP mineral resource estimate.*

CGA also supplied MA with 3 topographic surfaces: pre-mining topography, site topography at October 31st 2011, and "as-excavated" topography at October 31st 2011. With the consent of CGA, MA modified topographic surfaces as required.

CGA further supplied MA with models of underground workings in Colorado, Main Vein, Panique and Binstar.

14.3 Dimensions

The mineralised system as currently defined extends from 23,000 to 28,000 N (mine grid), 29,000 to 32,500 E and 600 to 1,300 mRL (Figure 14.1). It does not include the Pajo area to the north. This gives overall dimensions along strike of about of 6 km by 2 km at its widest point.

Each of the main mineralised bodies within the Masbate area was modelled separately according to surface vein orientation and drilling direction. In total, 12 separate areas were defined and modelled within 3 block models (Table 14.2, Figure 14.1).

Block 1	Colorado Grand View Montana
Block 2	Holy Moses Basalt
Block 3	Main Vein Libra Panique Binstar PH Panique Lacer Blue Quartz Boston Old Lady

Table 14.2 *Deposits modelled, 2011 MGP mineral resource estimate.*

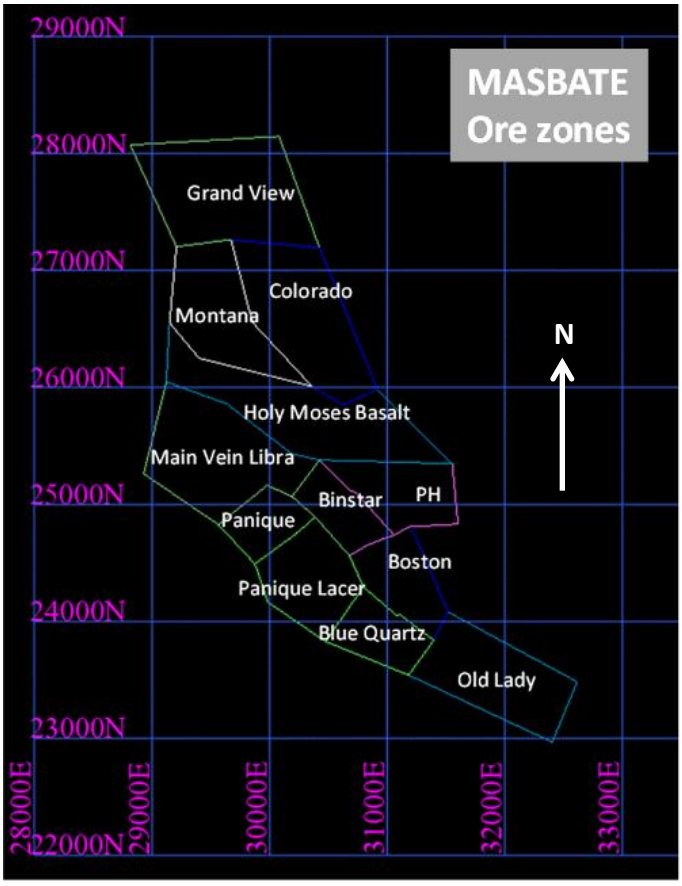


Figure 14.1 *Location of deposits, 2011 MGP mineral resource estimate. (MA 2012)*

Geological and grade modelling work encompassed all available drilling, including grade control. Modelling work was extended vertically to the limits of the current drill hole assay database.

14.4 Geologic Interpretation

3D geological modelling provides a model that is a representation of what is in the ground, derived from geological data that were captured and interpreted. As the information upon which it is based is not perfect, it cannot be an exact representation of reality, but can be a close approximation. The only time you will know exactly what was in the ground is when it is mined out. Before that the interpretation is formed from drill holes, trench samples, surface samples or mapping onto sections or plans.

A 3D geological model consists of the following:

- Drill holes in 3D space.
- A topographical surface.
- Any structural features e.g. faults.
- A volume of the mineralised body constructed from plans and sections.
- A block model with grades or other variables interpolated via geostatistics from the drill hole data.

Vein domain wireframes were constructed by the site geological team with input from MA. Drill hole sections were displayed on screen using Surpac, a geological modelling tool, from which vein intervals were picked, tagged into the database, and wireframes digitised directly from the

boundaries of the mineralised vein domains using a cut-off of > 0.4 Au g/t. The domains at times crossed, but drill intercepts were coded to just a single domain, and subsequent block similarly coded to just one domain, using a dominance hierarchy. There are a total of 112 currently defined vein domains with some containing multiple search orientations totalling a further 62 sub domains (Figure 14.2). The intervals for each drill hole for each domain were tagged into database tables.

The halo domain was interpreted on cross-sections and varied in direction and spacing to match the available drilling, to define the zone from 0.1 to 0.4 g/t Au. During this process, a low-grade assay contamination of about 0.2 g/t Au was noted in pre-2000 drilling and confirmed by statistical analysis. All pre-2000 drill holes have been removed from the informing sample data set, although they were used as a general guide, where appropriate, in the definition of the overall Halo domain. There are a total of 6 broad halo domains.

Further mineralised domains were defined for the Dumps (5), eluvial (7) and alluvial (1).

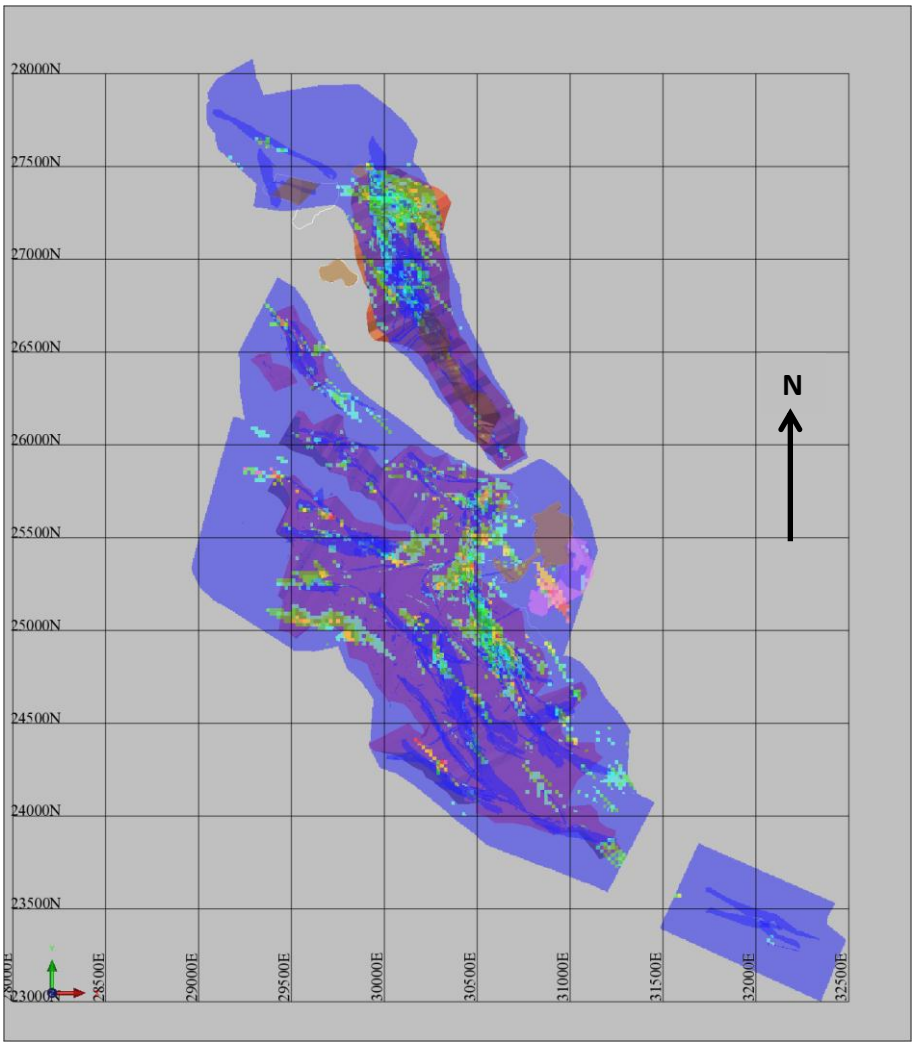


Figure 14.2 *Vein (dark blue) and Halo (red / orange) domain wireframes used within the 2011 MGP mineral resource estimate. Previous Halo domain (2008 MGP mineral resource estimate) illustrated in light blue. Note the significant reduction in the size of the 2011 Halo domain due to tighter boundaries and attention to low grade contamination from pre-2000 drilling data. (MA 2012)*

14.5 Data Preparation & Statistical Analysis

Statistical analysis of the grade data was carried out using Gemcom Surpac and MA in-house basic statistics. Surpac is an internationally recognised mine planning software toolbox which can be used at all stages of the mining process from initial feasibility studies though to production control. Surpac is currently used by numerous major global mining houses.

Prior to a statistical analysis grade domaining is normally required to delineate homogeneous areas of grade data. Statistical analysis does not take into account the spatial relationships of the data.

The purpose of the statistical analysis is to define the main characteristics of the underlying grade distribution to assist with the geological and grade modelling work. This process is important as the statistics of the individual sample populations can influence how the grade data is treated and the application of the grade estimation techniques. For example highly skewed data may require special grade capping and semivariogram analysis requirements.

The raw drill hole database is stored in an Access relational database. The Masbate Access database is connected directly to the Surpac for data display, down-hole compositing and wireframing of homogeneous grade domains.

14.5.1 Drill Hole Spacing

Drill hole data spacing is variable within each domain, although the Indicated resource areas are generally drilled on a 25 to 30 m section spacing.

14.5.2 Compositing

In 2006 mineral resource estimate, a 2 m composite length was chosen for the statistical analysis and grade estimation after looking at a range of composite lengths from 1 m to 5 m. The selection of 2 m composites took into consideration of grade statistics and the physical constraints of fitting composites into geometrically constrained vein domains. The 2 m composites were used for all domains. A 2 m composite length has been retained for this updated resource estimate.

The objective of compositing data is to obtain an even representation of sample grades and to eliminate any bias due to sample length (Volume Variance).

14.5.3 Stationarity

A domain is a three-dimensional volume that delineates the spatial limits of a single grade population, has a single orientation of grade continuity, and is geological homogeneous and has statistical and geostatistical parameters that are applicable throughout the volume (i.e. the principles of stationarity apply). Typical controls that can be used as the boundaries to the domains include structural features, weathering, mineralisation halos and lithology.

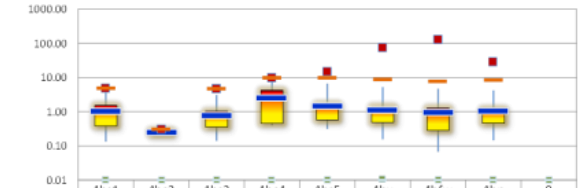
Due to the tight geological domaining, stationarity concerns are minimised with the resource estimation as each domain contains only one population of grade data. The mineralised domains are defined as high-grade where Au is greater than 0.4 g/t.

14.5.4 Basic Statistics

Basic statistics report the univariate statistical characteristics for each geological domain. The basic statistics are also used as a validation of the later resource estimates. The univariate statistics have been generated on each of the main domains at Masbate.

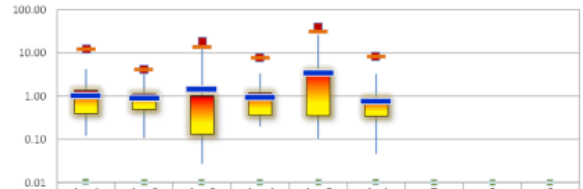
Data statistics for each domain and sub-domain are illustrated in Figure 14.3. Tables associated with the diagram for each domain highlight with the maximum grade, capped grade, mean grade, grade ranges and grade value for main percentiles. The statistics highlight the variability from area to area, resulting in the various top cuts (i.e. cap) used.

Binstar domains and sub-domains



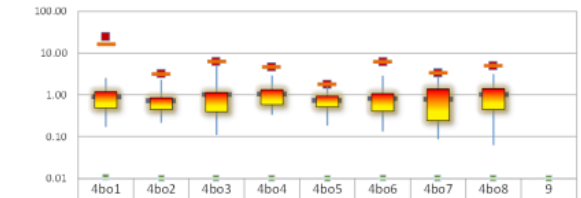
	4bs1	4bs2	4bs3	4bs4	4bs5	4bv	4bfw	4bn	9
25%	0.38	0.21	0.36	0.45	0.57	0.49	0.28	0.46	0.00
97.5%	3.60	0.30	3.10	9.13	6.60	5.25	4.65	4.16	0.00
2.5%	0.13	0.20	0.14	0.40	0.31	0.15	0.07	0.14	0.00
75%	1.54	0.28	1.04	4.25	1.67	1.40	1.29	1.20	0.00
mean	1.02	0.25	0.77	2.51	1.47	1.12	0.94	1.05	0.00
Max	4.83	0.31	4.66	9.83	14.78	75.33	129.49	28.57	0.00
Cap	4.83	0.31	4.66	9.83	9.79	8.73	7.77	8.43	0.00
Count	35	4	144	7	113	567	157	85	0
# Capped	0.0	0.0	0.0	0.0	2.0	9.0	1.0	1.0	0.0

Blue Quartz domains and sub-domains



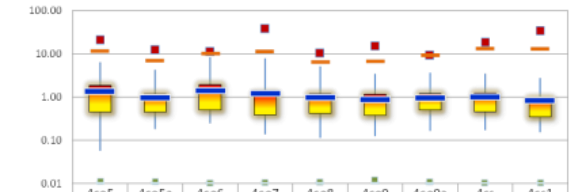
	4pq1	4pq2	4pq3	4pq4	4pq5	4pd	7	8	9
25%	0.39	0.49	0.13	0.36	0.35	0.34	0.00	0.00	0.00
97.5%	4.18	3.26	13.71	3.33	26.07	3.31	0.00	0.00	0.00
2.5%	0.12	0.11	0.03	0.20	0.10	0.05	0.00	0.00	0.00
75%	1.36	1.14	1.03	1.20	3.79	0.89	0.00	0.00	0.00
mean	1.03	0.90	1.45	0.93	3.43	0.76	0.00	0.00	0.00
Max	12.21	4.11	18.23	7.70	39.48	8.26	0.00	0.00	0.00
Cap	12.21	4.11	13.71	7.70	31.30	8.26	0.00	0.00	0.00
Count	403	109	46	61	48	216	0	0	0
# cut	0.0	0.0	2.0	0.0	1.0	0.0	0.0	0.0	0.0

Boston domains and sub-domains



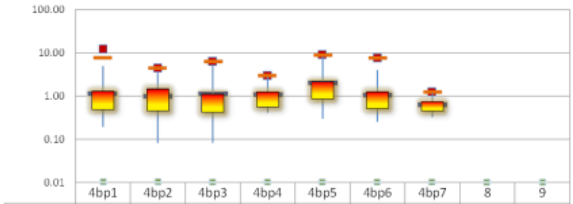
	4bo1	4bo2	4bo3	4bo4	4bo5	4bo6	4bo7	4bo8	9
mean	0.90	0.72	1.01	1.04	0.74	0.81	0.78	1.02	0.00
Max	24.79	3.22	6.28	4.64	1.78	6.15	3.36	4.91	0.00
Cap	16.30	3.14	6.28	4.64	1.78	6.15	3.36	4.91	0.00
25%	0.48	0.44	0.39	0.58	0.51	0.41	0.24	0.45	0.00
97.5%	2.54	2.31	4.96	2.93	1.59	2.88	2.81	3.16	0.00
2.5%	0.17	0.21	0.11	0.33	0.18	0.13	0.09	0.06	0.00
75%	1.20	0.85	1.14	1.34	0.94	1.08	1.40	1.42	0.00
Count	637	137	69	93	19	171	45	36	0
# Capped	1.0	3.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0

Colorado domains and sub-domains



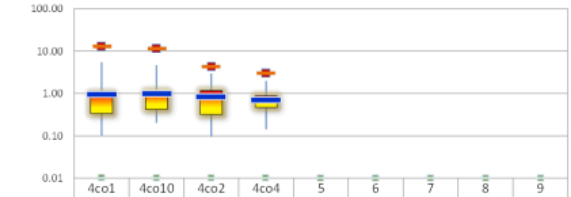
	4co5	4co5a	4co6	4co7	4co8	4co9	4co9a	4cs	4cs1
25%	0.45	0.45	0.50	0.38	0.42	0.38	0.50	0.45	0.34
97.5%	6.43	4.21	8.27	7.84	5.10	3.43	3.65	3.46	2.73
2.5%	0.06	0.18	0.24	0.14	0.11	0.12	0.16	0.17	0.15
75%	1.85	1.17	1.94	1.20	1.18	1.17	1.19	1.24	0.94
mean	1.34	0.96	1.41	1.20	0.97	0.87	0.95	1.00	0.82
Max	20.90	12.33	11.17	37.92	10.41	15.10	9.24	18.47	34.03
Cap	11.66	6.93	10.11	11.33	6.40	6.66	9.24	13.12	12.95
Count	1017	925	120	363	554	1854	718	235	158
# capped	2.0	1.0	2.0	8.0	9.0	2.0	0.0	1.0	1.0

Boston Porphyry domains and sub-domains



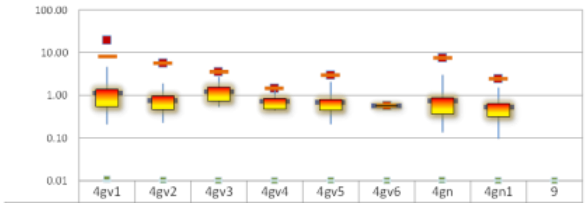
	4bp1	4bp2	4bp3	4bp4	4bp5	4bp6	4bp7	8	9
mean	1.12	0.99	1.12	1.07	2.01	1.04	0.63	0.00	0.00
Max	12.17	4.43	6.29	2.97	9.00	7.54	1.23	0.00	0.00
Cap	7.67	4.43	6.29	2.97	8.85	7.54	1.23	0.00	0.00
25%	0.48	0.45	0.42	0.56	0.85	0.51	0.45	0.00	0.00
97.5%	4.94	3.48	5.33	2.74	8.21	3.97	1.16	0.00	0.00
2.5%	0.19	0.08	0.08	0.41	0.29	0.25	0.32	0.00	0.00
75%	1.30	1.46	1.11	1.25	2.14	1.25	0.74	0.00	0.00
Count	242	165	31	10	21	99	23	0	0
# capped	1.0	0.0	0.0	0.0	1.0	0.0	0.0	0.0	0.0

Colorado (cont.) domains and sub-domains



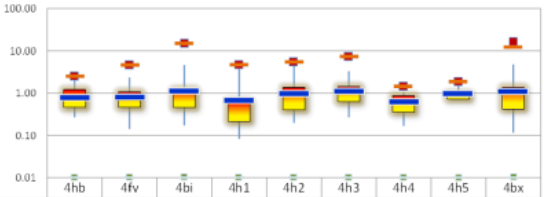
	4co1	4co10	4co2	4co4	5	6	7	8	9
25%	0.34	0.42	0.32	0.47	0.00	0.00	0.00	0.00	0.00
97.5%	5.46	4.66	2.94	1.97	0.00	0.00	0.00	0.00	0.00
2.5%	0.10	0.20	0.10	0.14	0.00	0.00	0.00	0.00	0.00
75%	1.02	1.13	1.17	0.89	0.00	0.00	0.00	0.00	0.00
mean	0.94	0.99	0.83	0.70	0.00	0.00	0.00	0.00	0.00
Max	12.81	11.49	4.30	2.98	0.00	0.00	0.00	0.00	0.00
Cap	12.81	11.49	4.30	2.98	0.00	0.00	0.00	0.00	0.00
Count	179	89	111	64	0	0	0	0	0
# Capped	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0

Grand View domains and sub-domains



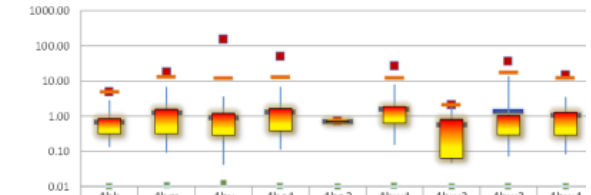
	4gv1	4gv2	4gv3	4gv4	4gv5	4gv6	4gn	4gn1	9
mean	1.13	0.76	1.22	0.72	0.68	0.57	0.74	0.53	0.00
Max	19.54	5.63	3.54	1.46	2.95	0.58	7.48	2.44	0.00
Cap	8.16	5.63	3.54	1.46	2.95	0.58	7.48	2.44	0.00
25%	0.55	0.47	0.74	0.48	0.45	0.56	0.37	0.32	0.00
97.5%	4.61	1.89	2.68	1.31	2.08	0.57	3.02	1.53	0.00
2.5%	0.21	0.23	0.53	0.44	0.21	0.56	0.14	0.10	0.00
75%	1.39	0.97	1.57	0.84	0.80	0.57	0.87	0.64	0.00
Count	839	288	36	11	100	2	295	63	0
# Capped	1.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0

Holy Moses - Basalt domains and sub-domains



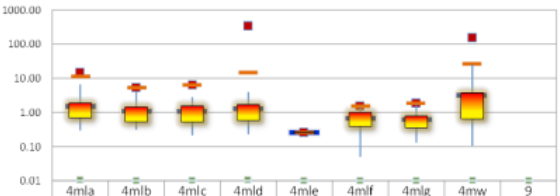
	4hb	4fv	4bi	4h1	4h2	4h3	4h4	4h5	4bx
25%	0.46	0.46	0.46	0.21	0.40	0.63	0.35	0.72	0.42
97.5%	2.17	2.34	4.62	3.98	5.22	3.30	1.06	1.75	4.81
2.5%	0.26	0.14	0.17	0.08	0.20	0.27	0.17	0.72	0.12
75%	1.23	1.07	1.23	0.65	1.37	1.43	0.87	0.99	1.38
mean	0.78	0.80	1.12	0.67	0.97	1.10	0.62	0.97	1.09
Max	2.51	4.62	15.07	4.68	5.50	7.32	1.44	1.86	16.44
Cap	2.51	4.62	15.07	4.68	5.50	7.32	1.44	1.86	12.17
Count	144	180	426	25	42	114	29	6	313
# Capped	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	1.0

Holy Moses - Basalt (cont.) domains and sub-domains



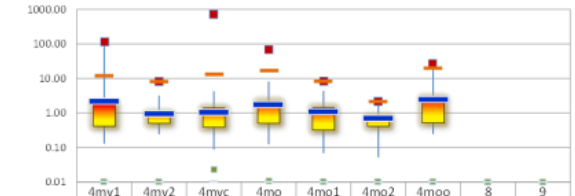
	4hh	4hm	4hv	4ha1	4ha2	4hw1	4hw2	4hw3	4hw4
mean	0.68	1.25	0.90	1.32	0.70	1.58	0.57	1.35	1.07
Max	4.92	17.98	153.11	50.17	0.73	27.02	2.11	36.56	14.83
Cap	4.92	13.13	12.02	12.88	0.73	12.32	2.11	17.45	12.23
25%	0.32	0.31	0.28	0.39	0.68	0.63	0.06	0.29	0.29
97.5%	2.83	6.80	3.60	6.88	0.73	7.92	1.89	13.78	3.51
2.5%	0.13	0.09	0.04	0.11	0.66	0.15	0.05	0.07	0.08
75%	0.86	1.59	1.22	1.67	0.72	1.89	0.83	1.06	1.31
Count	106	709	1852	178	3	320	9	47	48
# Capped	0.0	1.0	2.0	1.0	0.0	5.0	0.0	1.0	1.0

Main Vein domains and sub-domains



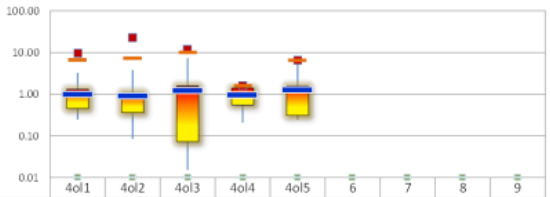
	4mla	4mlb	4mlc	4mld	4mle	4mlf	4mlg	4mw	9
mean	1.50	1.09	1.08	1.27	0.26	0.67	0.62	3.18	0.00
Max	14.58	5.27	6.38	340.16	0.26	1.53	1.84	153.28	0.00
Cap	11.34	5.27	6.38	14.66	0.26	1.53	1.84	26.15	0.00
25%	0.69	0.53	0.52	0.58	0.26	0.39	0.36	0.65	0.00
97.5%	6.60	4.08	2.81	3.94	0.26	1.49	1.77	26.15	0.00
2.5%	0.30	0.31	0.21	0.23	0.26	0.05	0.13	0.10	0.00
75%	1.89	1.46	1.56	1.69	0.26	1.01	0.78	3.77	0.00
Count	395	55	118	506	1	47	24	222	0
# Capped	1.0	0.0	0.0	1.0	0.0	0.0	0.0	6.0	0.0

Main Vein (cont.) domains and sub-domains



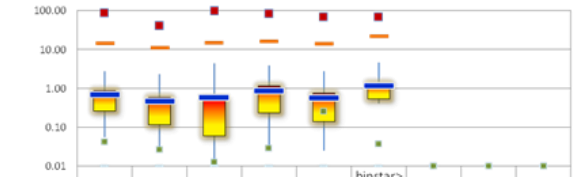
	4mv1	4mv2	4mvc	4mo	4mo1	4mo2	4moo	8	9
25%	0.41	0.50	0.39	0.50	0.32	0.41	0.51	0.00	0.00
97.5%	88.11	3.18	4.24	8.17	4.32	2.01	19.66	0.00	0.00
2.5%	0.13	0.24	0.09	0.12	0.07	0.05	0.24	0.00	0.00
75%	1.81	1.00	1.39	2.16	1.44	0.77	2.74	0.00	0.00
mean	2.15	0.94	1.03	1.73	1.08	0.70	2.44	0.00	0.00
Max	114.00	8.09	706.14	67.41	8.20	2.12	26.40	0.00	0.00
Cap	11.92	8.09	13.15	16.90	8.20	2.12	19.66	0.00	0.00
Count	20	86	7110	410	59	13	57	0	0
# Capped	2.0	0.0	10.0	5.0	0.0	0.0	2.0	0.0	0.0

Old Lady domains and sub-domains



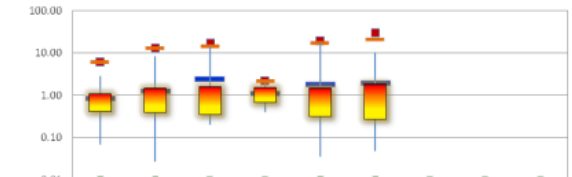
	4ol1	4ol2	4ol3	4ol4	4ol5	6	7	8	9
25%	0.45	0.36	0.07	0.56	0.31	0.00	0.00	0.00	0.00
97.5%	3.27	3.84	7.35	1.57	5.46	0.00	0.00	0.00	0.00
2.5%	0.25	0.09	0.02	0.21	0.25	0.00	0.00	0.00	0.00
75%	1.32	1.10	1.59	1.40	1.60	0.00	0.00	0.00	0.00
mean	1.00	0.90	1.22	0.96	1.27	0.00	0.00	0.00	0.00
Max	9.58	22.75	11.87	1.59	6.51	0.00	0.00	0.00	0.00
Cap	6.62	7.31	10.06	1.59	6.51	0.00	0.00	0.00	0.00
Count	133	107	22	7	12	0	0	0	0
# Capped	1.0	2.0	1.0	0.0	0.0	0.0	0.0	0.0	0.0

Panique domains and sub-domains



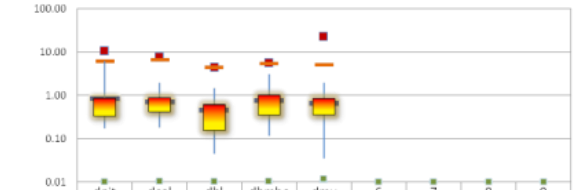
	gv	co	hmbw	hmbe	binstar	binstar> 0.4	7	8	9
25%	0.26	0.12	0.06	0.24	0.14	0.53	0.00	0.00	0.00
97.5%	2.75	2.32	4.43	3.86	2.74	4.62	0.00	0.00	0.00
2.5%	0.06	0.03	0.01	0.03	0.03	0.41	0.00	0.00	0.00
75%	0.88	0.59	0.55	1.16	0.75	1.36	0.00	0.00	0.00
mean	0.68	0.46	0.58	0.86	0.57	1.16	0.00	0.00	0.00
Max	86.90	41.15	98.35	83.09	68.39	68.39	0.00	0.00	0.00
Cap	14.36	11.15	14.85	16.17	14.07	21.95	0.00	0.00	0.00
# Capped	16.0	11.0	3.0	12.0	36.0	15.0	0.0	0.0	0.0
Count	15587	10540	2553	11507	35140	14166	0	0	0

Halo domains and sub-domains



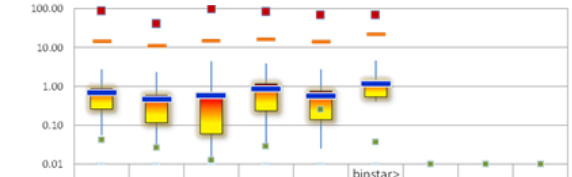
	4pl1	4pl2	4pl3	4pl4	4pl5	4pl6	7	8	9
mean	0.84	1.25	2.37	1.09	1.76	1.91	0.00	0.00	0.00
Max	6.05	12.87	16.81	2.16	19.08	29.60	0.00	0.00	0.00
Cap	6.05	12.87	14.26	2.16	17.13	20.74	0.00	0.00	0.00
25%	0.41	0.39	0.35	0.67	0.31	0.27	0.00	0.00	0.00
97.5%	2.82	8.30	14.26	2.08	17.13	10.12	0.00	0.00	0.00
2.5%	0.07	0.03	0.20	0.39	0.03	0.05	0.00	0.00	0.00
75%	1.08	1.46	1.61	1.51	1.48	1.84	0.00	0.00	0.00
Count	53	85	8	4	31	47	0	0	0
# capped	0.0	0.0	1.0	0.0	1.0	1.0	0.0	0.0	0.0

Dump domains and sub-domains



	dpit	dcol	dbl	dhmbe	dmv	6	7	8	9
mean	0.83	0.70	0.45	0.76	0.65	0.00	0.00	0.00	0.00
Max	10.56	7.70	4.42	5.59	22.83	0.00	0.00	0.00	0.00
Cap	6.15	6.61	4.42	5.43	5.09	0.00	0.00	0.00	0.00
25%	0.33	0.42	0.15	0.35	0.35	0.00	0.00	0.00	0.00
97.5%	6.15	1.94	1.45	3.07	1.95	0.00	0.00	0.00	0.00
2.5%	0.17	0.18	0.04	0.12	0.03	0.00	0.00	0.00	0.00
75%	0.86	0.88	0.62	1.02	0.85	0.00	0.00	0.00	0.00
# Capped	2.0	1.0	0.0	1.0	2.0	0.0	0.0	0.0	0.0
Count	72	388	250	333	1705	0	0	0	0

Grade Control domains and sub-domains



	gv	co	hmbw	hmbe	binstar	binstar> 0.4	7	8	9
25%	0.26	0.12	0.06	0.24	0.14	0.53	0.00	0.00	0.00
97.5%	2.75	2.32	4.43	3.86	2.74	4.62	0.00	0.00	0.00
2.5%	0.06	0.03	0.01	0.03	0.03	0.41	0.00	0.00	0.00
75%	0.88	0.59	0.55	1.16	0.75	1.36	0.00	0.00	0.00
mean	0.68	0.46	0.58	0.86	0.57	1.16	0.00	0.00	0.00
Max	86.90	41.15	98.35	83.09	68.39	68.39	0.00	0.00	0.00
Cap	14.36	11.15	14.85	16.17	14.07	21.95	0.00	0.00	0.00
# Capped	16.0	11.0	3.0	12.0	36.0	15.0	0.0	0.0	0.0
Count	15587	10540	2553	11507	35140	14166	0	0	0

Figure 14.3 *Univariate statistics for deposit domains and sub-domains, 2011 MGP mineral resource estimate. (MA 2012)*

14.5.5 Grade Capping

Capping is the process of reducing the grade of the outlier sample to a value that is representative of the surrounding grade distribution. Reducing the value of an outlier sample grade minimises the overestimation of adjacent blocks in the vicinity of an outlier grade value. At no stage are sample grades removed from the database if grade capping is applied.

Table 14.3 displays the grade ranges which were applied for each mineralised zone along with the maximum grade and number of samples which were capped during the making of the block model.

Mineralised Zone	No. of Domains	No. Domains capped	Maximum Au	Range of Capping Grade	No. Samples Capped
Binstar	8	4	129.49	7.77 - 9.79	13
Blue Quartz	6	2	39.48	13.71 - 31.30	3
Boston	15	4	24.79	3.14 - 16.3	6
Colorado	13	8	37.92	6.4 - 13.12	26
Dumps	5	4	22.83	5.09 - 6.61	6
Grand View	8	1	19.54	8.16	1
Holy Moses	18	7	153.11	12.02 - 17.45	12
Main Vein	15	7	706.14	11.34 - 26.15	27
Old Lady	5	3	22.75	6.62 - 10.06	4
Panique	6	3	29.6	14.26 - 20.74	3
Vein - GC			166.4	15	219
Halo - drill			36.36	3.5	72
Halo - GC			190.43	3.5	320

Table 14.3 *Capping information, 2011 MGP mineral resource estimate.*

14.6 Variography

The most important bivariate statistic used in geostatistics is the semivariogram. The experimental semivariogram is estimated as half the average of squared differences between data separated exactly by a distance vector 'h'. Semivariogram models used in grade estimation should incorporate the main spatial characteristics of the underlying grade distribution at the scale at which mining is likely to occur.

The semivariogram analysis was undertaken for individual elements within each major grade domain that contain sufficient data to allow a semivariogram to be generated. Three dimensional (3-D) semivariograms are generated using three orthogonal principal directions.

14.6.1 Methodology

For each variable in each domain, the experimental variogram containing the clearest structure and greatest difference in range between each direction was selected for use in model fitting where possible. The variogram modelling process using variables is described as follows:

- Experimental variograms with small lags orientated down hole to aid interpretation of nugget effect
- Omni-directional variogram to determine optimal lag distance for directional component of variogram
- Variogram map, computing 18 directions in the reference plane and normal to the reference plane
- Directional variogram with 2 directions in reference plane (down dip) oriented parallel to the average orientation of the wireframe models of each domain, plus variogram normal to the plane (across strike).

14.6.2 Variogram Models & Parameters

Due to the large number of vein domains, and broad nature of the halo domain, a single variogram model was used for each domain type. Search directions and ranges were varied for each domain, as noted earlier. Variogram parameters are documented in Table 14.4.

Model	Type	Parameters
Halo (Figure 14.4)	3 structure nested spherical	nugget 0.06, c1= 0.09, a1= 10, c2 =0.01, a2=50, c3=0.35, a3= 250
Vein (Figure 14.5)	2 structure nested spherical	nugget 0.35, c1= 0.52, a1= 12, c2 =0.27, a2 =200
Dumps & Eluvial (Figure 14.6)	omnidirectional	nugget 0.2, c1=0.55 a1=50

Table 14.4 *Variogram parameters, 2011 MGP mineral resource estimate.*

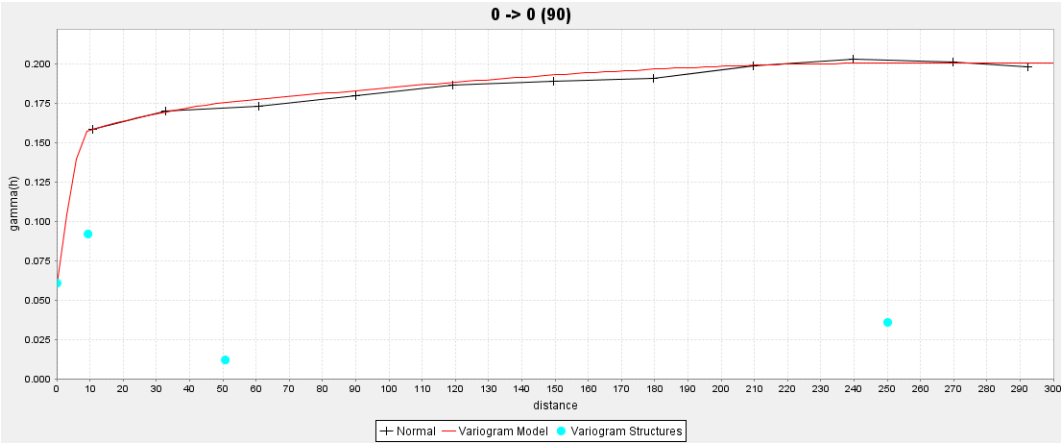


Figure 14.4 *Data variogram for Halo, major axis, 2011 MGP mineral resource estimate. (MA 2012)*

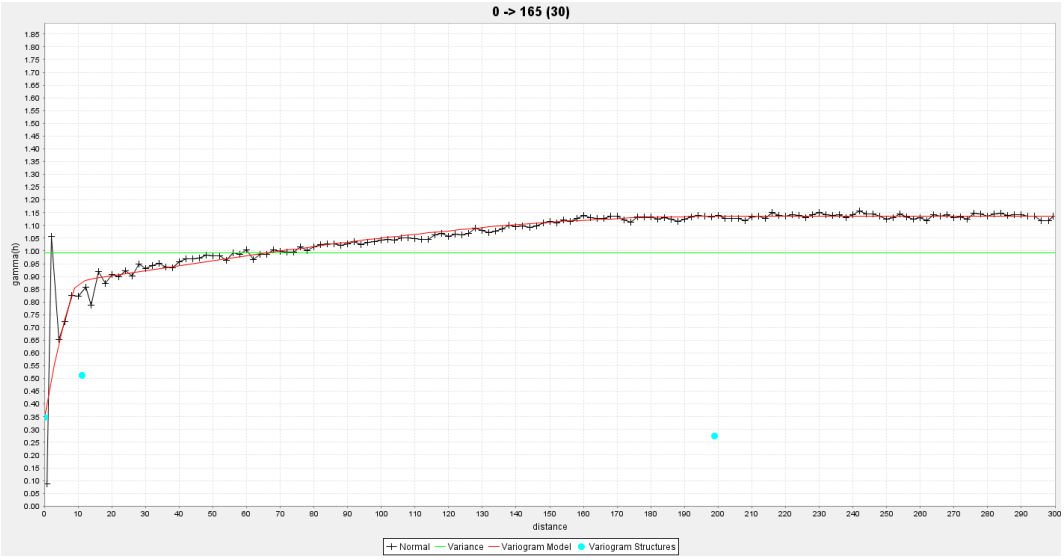


Figure 14.5 *Data variogram for Vein, major axis, 2011 MGP mineral resource estimate. (MA 2012)*

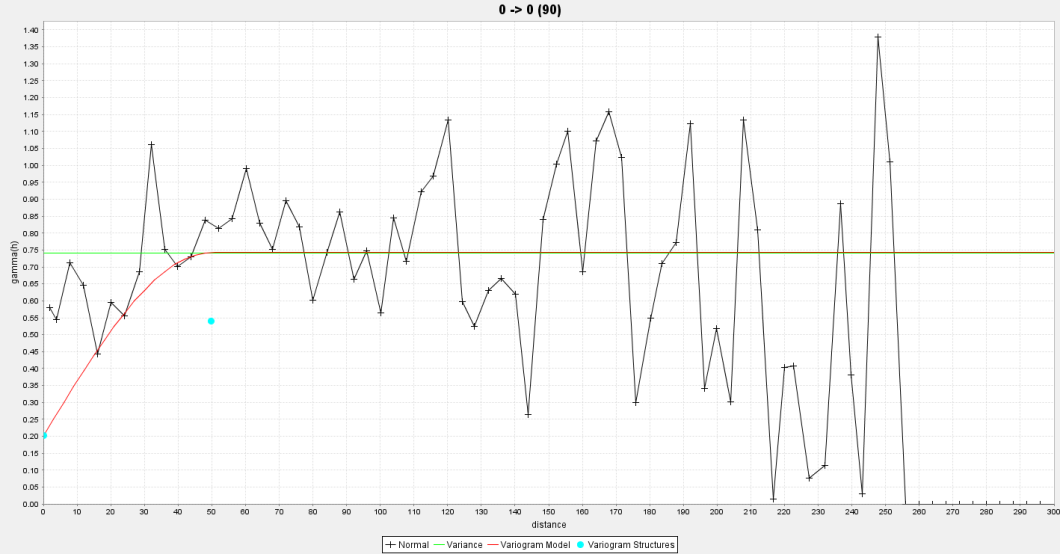


Figure 14.6 *Data variogram for Dumps and Eluvial, omnidirectional, 2011 MGP mineral resource estimate. (MA 2012)*

14.7 Grade Estimation

14.7.1 Methodology

The grade was estimated using Ordinary Kriging, and then checked against the informing sample statistics as well as estimates using nearest neighbour (3D polygonal) and inverse distance squared estimates.

14.7.2 Ordinary Kriging

Ordinary kriging ("OK") is a robust grade estimation technique and is the main algorithm used in geostatistics. The most common use of OK is to estimate the average grades into panels at the scale of the available drill hole spacing. OK is a globally unbiased estimator which produces the least error variance for grade estimates.

It uses the grade continuity information from the semivariogram to estimate grades into panels. It is also able to accommodate anisotropy within the data and is able to replicate this in the panel estimates. Another important feature of kriging is that it automatically deals with clustering of data which is important in areas where the data density is not uniform.

Due to variable non-stationary issues OK was used in preference to simple kriging ("SK") as only local stationarity conditions are required for the OK algorithm.

Kriging forms a sound basis for generating panel grade estimates at a scale which is appropriate to the sample density. If tonnes and grades are required for volumes smaller than the kriging panel size then other more advanced non-linear techniques need to be applied.

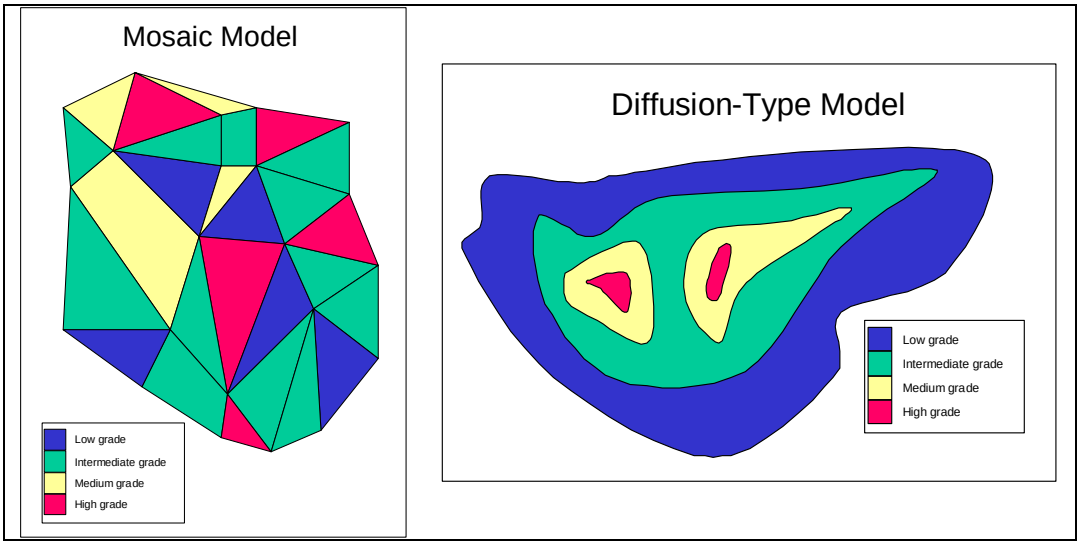


Figure 14.7 *The two end member models of grade correlation within a domain. (MA 2012)*

Grades within the diffusion model show a systematic gradational relationship (Figure 14.7). This model is appropriate if low grades tend to be adjacent to intermediate grades, which tend to be adjacent to high grades. The transition between high, medium and low grades is generally systematic resulting in edge-effects. The different grade zones are assumed to be correlated in as much as going from low to high grades requires the transitional grades to be medium.

The 'mosaic model', consists of random 'parcels' of mineralisation with grades that show no obvious systematic trend (Figure 14.7). High-grade zones can occur adjacent to waste zones with any combination of adjacent grade zones possible and no obvious correlation between the various 'parcels'. The correlation between adjacent zones is very low. In this case we say that there are no edge-effects.

If the diffusion model best represents the mineralisation being examined, a linear estimation method such as Ordinary Kriging or non-linear estimation method such as a gaussian estimation methodology for recoverable resource estimation.

If the mosaic model best describes the grade continuity, it is assumed that a minimum correlation occurs between the mineralised grade zones within the broad grade envelope and an indicator based estimation methodology is preferred.

The mineralisation at Masbate most closely follows the diffusion model.

14.7.3 Block Model & Panel Size

The dimensions and parameters for the 3D model are shown below in Table 14.5. The dimensions for the block models were chosen using the extents of the original drill hole data and considering the orientation of the veins and halo domain wireframes: longer in the north-south direction, narrower in the east-west direction, with the depth of the blocks determined by mining bench height. The estimation block size was varied with the resource category, which reflects the level of confidence, which in turn is dependent on the informing sample spacing and variance.

When filling the 3D block model, the mineralised domain wireframes were used as a constraint, so only blocks within that solid were filled. The informing data sets were not constrained, there being a separate one for each domain.

Type	Y	X	Z
Minimum Coordinates	22980	28920	600
Maximum Coordinates	28100	32560	1360
Estimation Block Size	40	40	20
Min. Block Size	5	5	2.5
Rotation	0	0	0
For Indicated Category Estimates			
Estimation Block Size	20	20	10
For Measured Category Estimates			
Estimation Block Size	10	10	5

Table 14.5 *Block Model dimensions, 2011 MGP mineral resource estimate.*

14.7.4 Search Parameters

The search dimensions have the two longer values in the plane of the mineralisation, with the first direction in the horizontal plan and the second longer value in the vertical direction. The shorter search direction is in the cross-strike orientation perpendicular to the plan of the approximately vertical dipping vein mineralisation.

A search of 200 m (N) in the major axis was used for all domains except halo, which used 250 m. The major axis was set along strike, with the semi-major axis being the same as the major and minor axis ration being 4. Search parameters have been orientated parallel to the main strike and dip of the mineralization.

A single pass was used to fill each of the domains. Unfilled blocks within the wireframes are minor.

14.7.5 Informing Samples

The minimum number of informing samples was varied from 3 to 32 by resource category as follows:

- Inferred resource category - minimum of 3 informing samples.
- Indicated resource category – minimum of 8 informing samples
- Measured resource category – minimum of 32 informing samples

Estimates were made using Ordinary Kriging, with octants further to restrict the influence of grade control drilling immediately below current mine floor. The maximum number of samples per octant was set to 4.

14.7.6 Search Radii

Search radii were found to be optimal at or near the distance that the variogram reached the sill. Thus the variogram ranges will be utilised in the maximum search distances. The isotropy apparent in the variogram analysis will be reflected in the search ellipse.

The search radii were set to same as the major range, being 250m for the halo domain, and 200m for all other domains.

14.7.7 Discretisation

Discretisation is used as a means to the estimate for the volume-variance effect by segmenting a block into pseudo points. The distance between a sample and a block is found by measuring both the distance and the direction between a sample location and the pseudo points. The variogram values between the sample and pseudo points are recorded as opposed to a singular variogram value for the sample to block distance. These variogram readings can then be averaged to provide a variogram value for the distance and direction between the sample and the block.

The dicretisation used was 3 x 3 x 3

14.7.8 Block Model Attributes

Attribute Name	Type	Decimals	Background	Description
au_ppm	Real	3	0	gold ppm by Ordinary Krig Estimation
code_area	Character	-	und	codes for areas, binstar,boston,colorado,gv,hmb,mont,mvl,ol,panique
code_domain	Character	-	und	codes for estimation domains, halo or see list veins, dumps, elv and alv
code_rock	Character	-	air	3 letter code: air,rck,dug,fll,dmp,elv,alv,vod, vn or hlo
est_avg_dist	Real	2	0	Anisotropic average distance to informing samples
Est_krg_var	Float	3	-99	estimation krig variance
est_near_hole	Character	-	und	name of nearest drill hole used in resource
est_numsamp	Integer	-	0	number of informing samples
est_slope	Real	-	0	estimation slope of regression
rescat	Character	-	und	mea, ind, inf, dug
sg	Real	2	0	in-situ dry bulk denisty - fsh=2.55, ox=2.37, sox=2.4 except colorado,gv,mont=2.55, dmp,elv,alv & fll=1.5
voids	Real	2	0	fraction inside voids, from 0 to 1
weathering	Character	-	air	air, ox, sox, fsh, dmp, alv, elv, fll

Table 14.6 *Block Model attributes, 2011 MGP mineral resource estimate.*

14.7.9 Voids

A number of historical underground workings are present within several MGP deposits (Colorado, Main Vein, Panique, Binstar). CGA provided MA a 3 dimensional model of the underground voids, constructed using historical records.

An allowance for underground voids has been included in the resource estimate, based on the proportion of individual resource block model cells which intersect the void model (recorded as a number between 0 and 1 for each block). A threshold test was conducted in the Main Vein area using complex void outlines and the resource model sub-block size of 5 x 5 x 2.5 m (XYZ). Resource model blocks were then selected for categorisation based on proportion cut-offs between 0.15 and 0.4 (Figure 14.8, Table 14.7).

A threshold value of 0.30 was selected for the final resource model estimate. Above the threshold value, the entire block is categorised as void, with estimated grade value re-set to 0.00 g/t, sg value re-set to 0.00 t/m^3 and weathering code re-set to “air” (i.e. underground workings are assumed to be open). Below the threshold value, the entire block retains its original attributes.

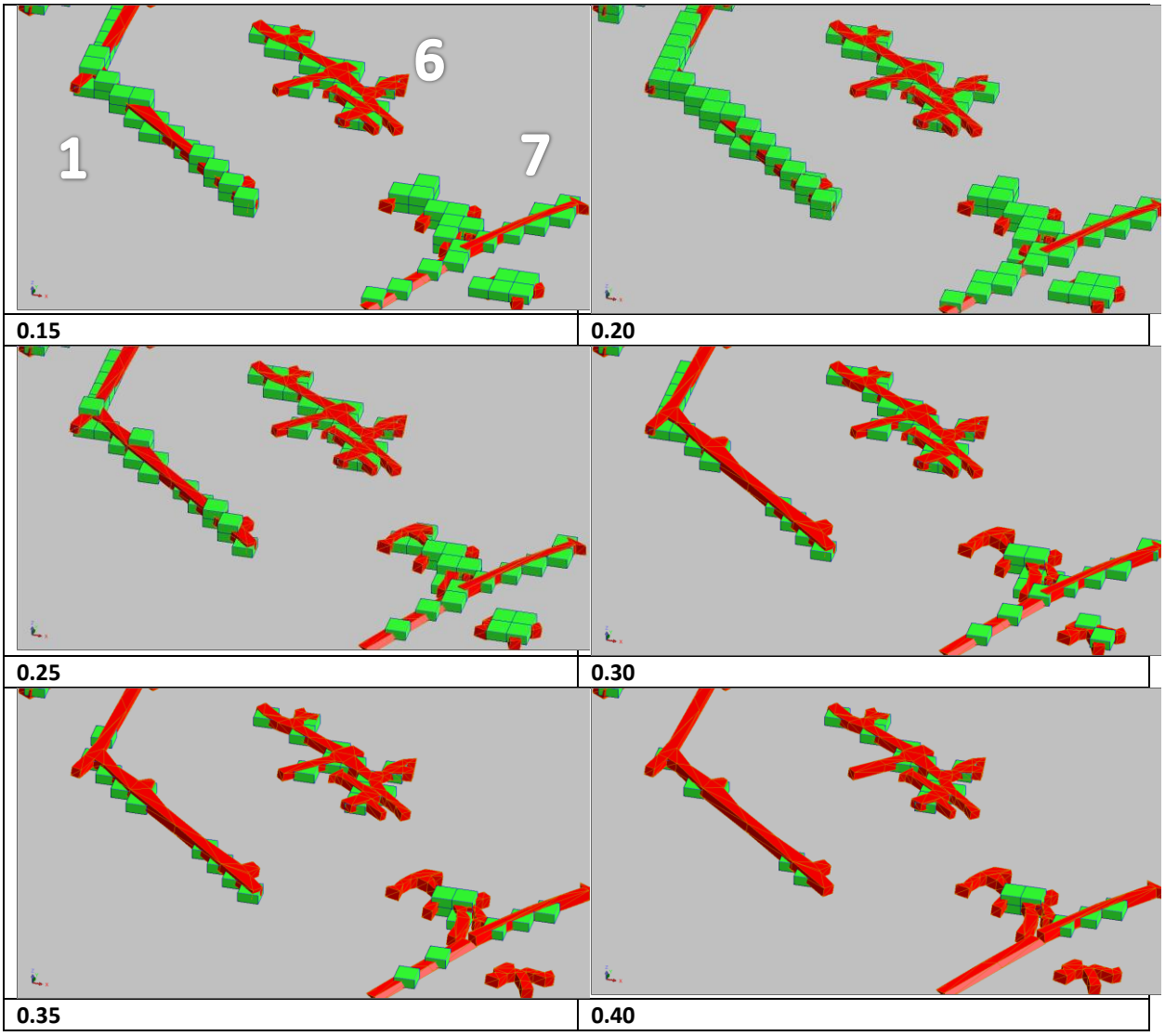


Figure 14.8 *Visual results of threshold testing for void categorisation. Red = underground void model for areas denoted 1, 6 and 7. Green = resource block cells selected for void category based on the proportion of cell which intersects the void. (MA 2012)*

Void wireframe	1	6	7
Void volume	**893**	**713**	**468**
Threshold Proportion			
0.25	1,312.5	1,125	750
0.3	875	1,000	437.5
0.35	625	750	375
0.4	312.5	687.5	312.5

Table 14.7 *Volume comparison between selected void areas (1, 6 and 7 of Figure 14.8) and resource block model cells at various proportion. A threshold proportion of 0.3 was selected for void categorization in the final resource estimate.*

14.8 Validation & Comparison to Previous Estimates

The 2008 resource estimate *NI43-101 Technical Report on the mineral resources of the Masbate Deposit, Masbate Province, Republic of the Philippines,* incorporated 108 new holes plus the existing

database to define new vein & stockwork for the Boston and Libra mineralised zones, and an enveloping low-grade halo mineralisation for all areas. The grade estimation used ordinary kriging for the halo and inverse distance squared for vein and stockwork into a parent block size of 20 m (E) by 20 m (N) by 10 m (RL). A sub-block size of 5 m (E) by 5 m (N) by 2.5 m (RL) was used against all wireframes for volumes. No uniform conditioning was used as the mining scenario has now changed from selective to bulk mining.

The report produced a revised resource estimation reporting indicated and inferred resources at various cut-off grades. The resources for each of the project areas was estimated and combined into a single summary table. The resources were divided between the hard rock (Table 14.8) and dumps (Table 14.9).

COMBINED MASBATE		Cut off grade															
		0.0	0.1	0.2	0.3	0.36	0.4	0.5	0.6	0.7	0.8	0.9	1.0	1.2	1.5	1.7	2.0
Indicated	Au g/T	0.59	0.60	0.68	0.85	0.96	1.02	1.12	1.30	1.56	1.79	1.91	1.98	2.11	2.37	2.59	2.93
	M Tonnes	276.00	268.71	228.25	164.58	135.33	124.06	105.01	79.16	56.77	44.62	39.67	36.84	32.19	23.82	18.67	12.78
	Au M oz	5.24	5.22	5.01	4.50	4.19	4.05	3.78	3.32	2.86	2.57	2.43	2.35	2.18	1.82	1.55	1.20
Inferred	Au g/T	0.40	0.42	0.48	0.63	0.79	0.88	1.06	1.29	1.62	1.94	2.14	2.30	2.52	2.88	3.13	3.50
	M Tonnes	420.46	401.67	328.83	196.33	127.15	103.69	73.80	50.40	33.50	24.60	20.77	18.31	15.43	11.86	9.91	7.72
	Au M oz	5.45	5.40	5.04	3.96	3.22	2.94	2.51	2.09	1.74	1.53	1.43	1.35	1.25	1.10	1.00	0.87

Table 14.8 *2008 MA mineral resource estimate, combined deposits, MGP. Sourced from Vigar (2008).*

DUMPS		Cut off grade															
		0.0	0.1	0.2	0.3	0.36	0.4	0.5	0.6	0.7	0.8	0.9	1.0	1.2	1.5	1.7	2.0
Indicated	Au g/T	0.61	0.61	0.61	0.62	0.62	0.63	0.67	0.74	0.80	0.89	0.95	1.03	1.22	0.00	0.00	0.00
	M Tonnes	18.65	18.57	18.53	18.35	18.08	17.75	13.71	9.17	5.24	2.03	0.75	0.09	0.00	0.00	0.00	0.00
	Au M oz	0.37	0.37	0.37	0.36	0.36	0.36	0.30	0.22	0.13	0.06	0.02	0.00	0.00	0.00	0.00	0.00

Table 14.9 *2008 MA mineral resource estimate, pre-existing dumps before current mining operations commenced, MGP. Sourced from Vigar (2008).*

MASBATE + DUMPS		Cut off grade															
		0.0	0.1	0.2	0.3	0.36	0.4	0.5	0.6	0.7	0.8	0.9	1.0	1.2	1.5	1.7	2.0
Indicated	Au g/T	0.59	0.60	0.68	0.83	0.92	0.97	1.07	1.25	1.50	1.75	1.89	1.98	2.11	2.37	2.59	2.93
	M Tonnes	294.65	287.28	246.77	182.93	153.41	141.81	118.72	88.33	62.01	46.65	40.42	36.93	32.19	23.82	18.67	12.78
	Au M oz	5.60	5.59	5.38	4.86	4.55	4.41	4.07	3.54	2.99	2.62	2.46	2.35	2.18	1.82	1.55	1.20
Inferred	Au g/T	0.40	0.42	0.48	0.63	0.79	0.88	1.06	1.29	1.62	1.94	2.14	2.30	2.52	2.88	3.13	3.50
	M Tonnes	420.46	401.67	328.83	196.33	127.15	103.69	73.80	50.40	33.50	24.60	20.77	18.31	15.43	11.86	9.91	7.72
	Au M oz	5.45	5.40	5.04	3.96	3.22	2.94	2.51	2.09	1.74	1.53	1.43	1.35	1.25	1.10	1.00	0.87

Table 14.10 *2008 MA mineral resource estimate, total resource (combined deposits + dumps), MGP. Sourced from Vigar (2008).*

There was an additional historical eluvial resource sampled by Filminera in 2005 at Binstar and Colorado which had at that time not been reviewed. It was estimated to contain an inferred resource of: 412,753 t at 1.77 g/t Au for a total of 23,518 oz gold.

A comparison of these estimates with the 2011 estimates at a cut-off grade of 0.36 g/t Au shows an increase in tonnage and total ounces and decrease in grade over the 2008 estimates (Table 14.11), but remaining gold ounces has increased slightly, even after two years of production.

2008				2011			
Resource Category	Tonnage	Au grade	Contained Au	Resource Category	Tonnage	Au grade	Contained Au
	Mt	g/t	M oz		Mt	g/t	M oz
				Measured	6.4	0.77	0.16
				Indicated	197.8	0.76	4.84
Indicated	153.41	0.92	4.55	M&I	204.3	0.76	5.00
Inferred	127.15	0.79	3.22	Inferred	102.8	0.86	2.83

Table 14.11 *Comparison between 2008 and 2011 MGP mineral resource estimate.*

This may also be broken down by rock-type as follows, again above a cut-off of 0.36 g/t Au

Table 2: 2011 Mineral Resource Estimate break down by rock type					
Rock type	Measured	Grade	Indicated	Grade	Inferred
	tonnes	g/t Au	tonnes	g/t Au	tonnes
Vein	3,330,056	0.99	105,781,517	1.00	87,647,226
Halo	3,086,010	0.53	80,133,004	0.46	15,154,163
Dump			11,185,577	0.61	
Eluvial			711,849	0.93	
Alluvial			35,250	0.49	
Total Above	6,416,066	0.77	197,847,197	0.76	102,801,389

Table 14.12 *Breakdown by domain type, 2011 MGP mineral resource estimate.*

The halo mineralisation is showing a much lower grade than the vein mineralisation as would be expected.

A comparison of different estimation methods for the major vein domains was made using the large inferred size blocks (40 m x 40 m x 20 m). MA would normally extend this to the total resource as a validation exercise, but the large number of estimation domains and variable estimation block size made this an unrealistic approach for Masbate. The grade tonnage curves show different shapes, in particular due to the lack of smoothing and extreme selectivity (which of course cannot be achieved) in the nearest neighbour model, but similar total contained ounces overall.

At a cut-off of 0.4 g/t Au, the major vein domain estimates may be summarised as follows (Table 14.13, Figure 14.9):

Estimation Method	**Results**
Nearest Neighbour	157Mt @ 1.13 for 5.7 Moz
Inverse Distance Squared	182Mt @ 0.93 for 5.5 Moz
Ordinary Kriging	197 Mt @ 0.92 for 5.8 Moz

Table 14.13 *Comparison of major vein domains using different grade estimate techniques, cut-off grade = 0.4 g/t Au, 2011 MGP mineral resource estimate.*

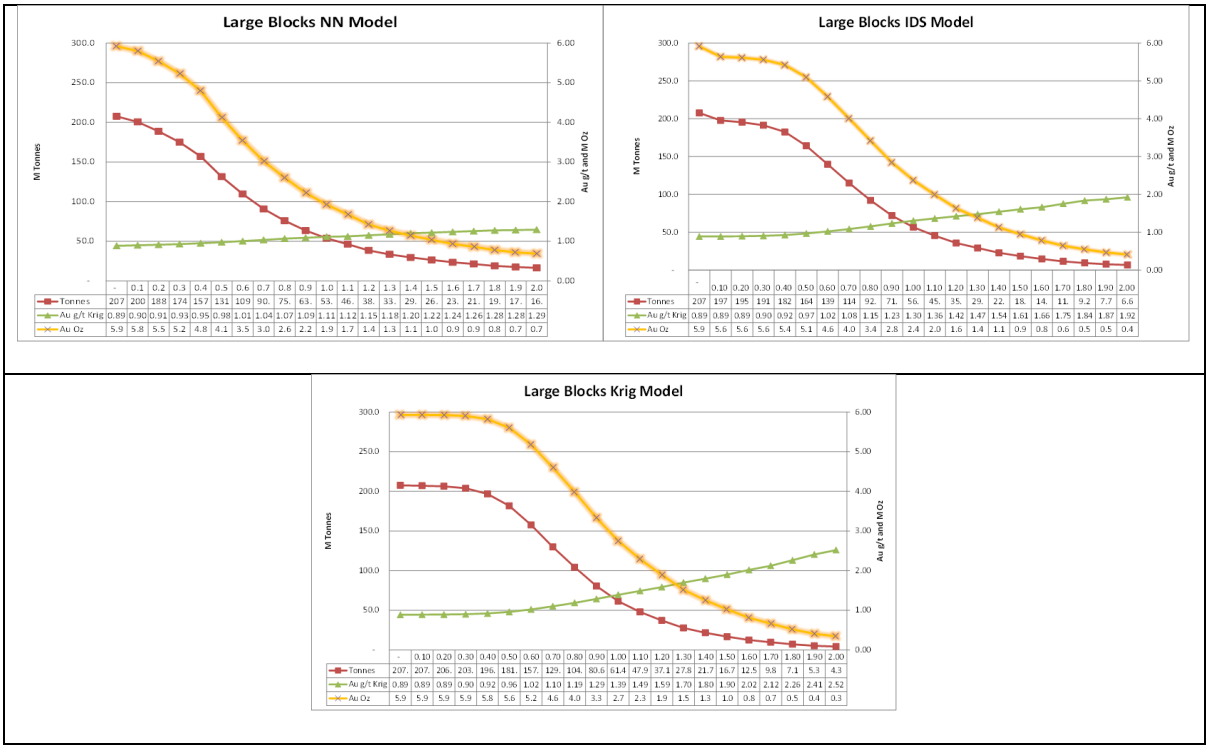


Figure 14.9 *Grade – tonnage curves for the major vein domains using large block size (40 m x 40 m x 20 m) and different grade estimate techniques. (MA 2012)*

A Trend Analysis was also conducted to compare the average grades from the informing sample data for the combined vein domains versus the krig block estimates. Due to the complex nature of the models, and the extreme clustering of the grade control data, the comparison shown in the graphs below is based on a comparison of the nearest neighbour and ordinary krig estimates by northing and elevation (Figure 14.10). The overall comparison is good, when we consider that the raw samples tend to give a slightly higher grade due to the lack of smoothing and lack of selectivity correction.

The grade with depth trend is of particular importance, as there may be a slight drop but no evidence of a major decrease in grade with depth. The sudden drop below -260 m RL may be due to a lack of data.

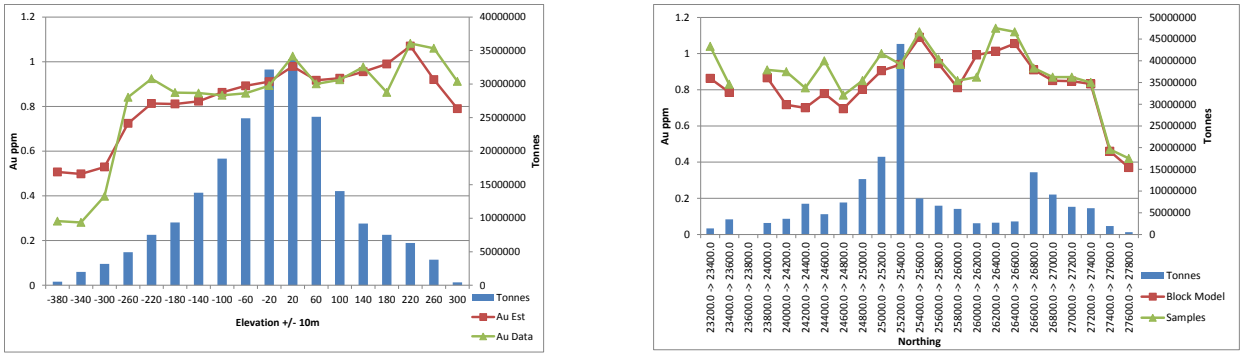


Figure 14.10 *Trend Analysis between average grade of informing samples (Green line) and resource block model grade estimates, vein domains. Left Hand = by depth. Right Hand= by Northing. 2011 MGP mineral resource estimate. (MA 2012)*

14.9 Economic Cut-off Parameters

All resources have been stated above a 0.36 g/t Au cut-off, which is supported as a suitable break between 0.3 and 0.4 g/t Au for this mineralised system.

The grade tonnage chart (Figure 14.11) indicates a sharp increase in both tonnage and ounces below 0.3 g/t Au. It should be remembered that the pre-2000 data was not used in the halo domain, where the majority of this low grade is located.

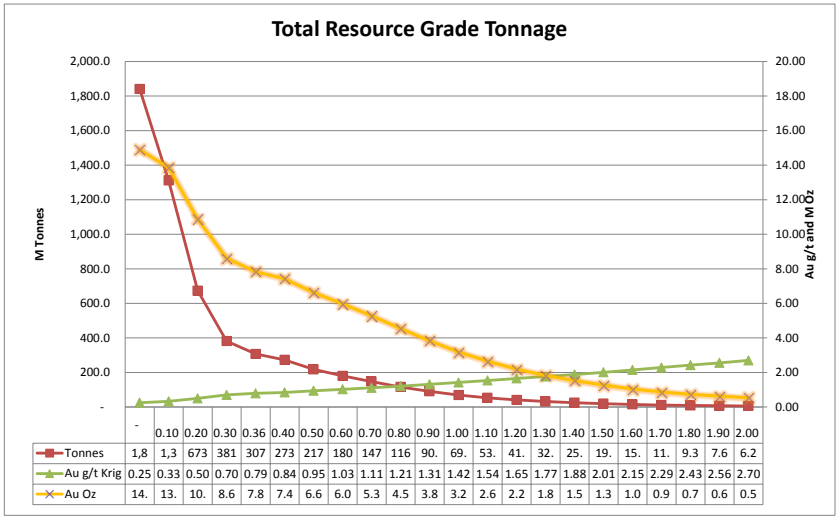


Figure 14.11 *Grade – tonnage curve, all domains, 2011 MGP mineral resource estimate. (MA 2012)*

14.10 Bulk Density

The bulk density values used for tonnage estimates are summarised in Table 14.14.

Area	Oxide	Transition	Fresh
Colorado	2.37	2.40	2.55
Main Vein	2.37	2.55	2.55
Holy Moses Basalt	2.37	2.55	2.55
Montana	2.37	2.40	2.55

Table 14.14 *Bulk density values, 2011 MGP mineral resource estimate.*

14.11 Moisture

No measurements were recorded.

14.12 Mining Factors

No mining factors have been applied to the in situ grade estimates for mining dilution or loss as a result of the grade control or mining process.

14.13 Metallurgical Factors

No metallurgical factors have been applied to the in situ grade estimates.

14.14 Assumptions for reasonable prospects for eventual economic extraction

Assumptions for reasonable prospects for eventual economic extraction applied to this deposit, include but may not be limited to the following[1]:

- Gold pricing at USD 1,000 per oz.
- Mining method will be by open cut.
- Processing method is CIL/CIP and heap leach.
- Processing costs of USD 7.50 per tonne.

Mineral resources estimated for this deposit are located within the physical boundaries of the MGP and are not subject to any known environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or any other factor which would impact on their potential economic extraction.

14.15 Resource Classification

Based on the study herein reported, delineated mineralisation of the Masbate Gold Project is classified as a resource according to the definitions from JORC Code standards:

> A 'Mineral Resource' is a concentration or occurrence of material of intrinsic economic interest in or on the Earth's crust in such form, quality and quantity that there are reasonable prospects for eventual economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge. Mineral Resources are sub-divided, in order of increasing geological confidence, into Inferred, Indicated and Measured categories. (JORC Code 2004)

A breakdown of the Masbate Project resource estimate by resource category is provided below in Table 14.15 and illustrated in Figure 14.12.

Resource Category	Tonnage	Au Grade	Contained Au
	Mt	g/t	M oz
Measured	6.4	0.77	0.16
Indicated	197.8	0.76	4.84
Total M & I	204.3	0.76	5.00
Inferred	102.8	0.86	2.83

Table 14.15 *2011 MGP Mineral Resource Estimate for combined domains, 0.36 g/t Au cut-off.*

For the classification of Mineral Resources for the Masbate Project, the following definitions were adopted and applied to each domain separately;

- Inferred resource category - minimum of 3 informing samples.
- Indicated resource category – minimum of 7 informing samples and Krig Slope of 0.3.
- Measured resource category – minimum of 32 informing samples and within 10m of grade control drilling

[1] Figures documented have been used by MA to form an opinion on the potential economic viability of extraction of the mineral resource. They are not the figures utilised to generate the reserves of the MGP.

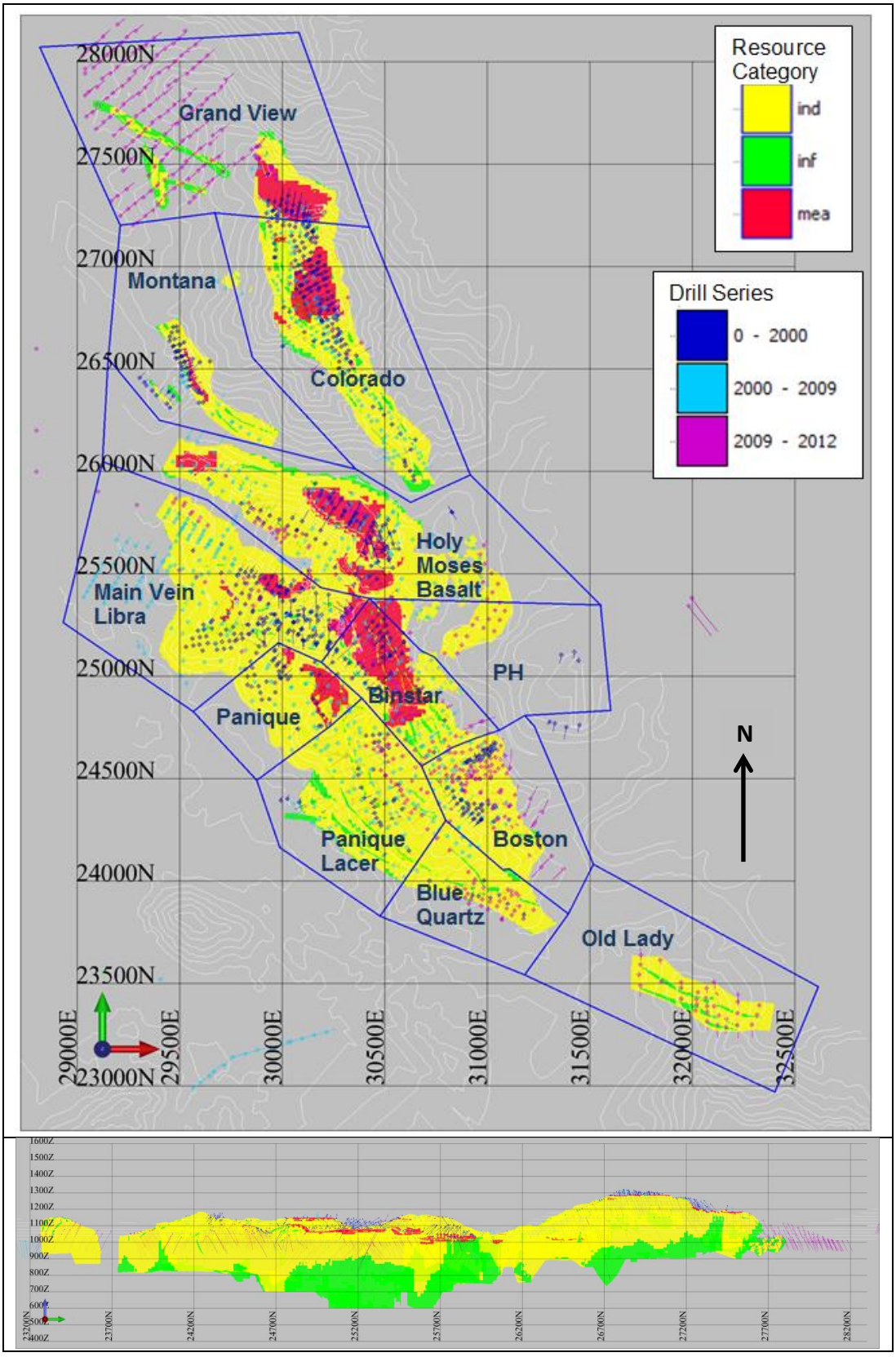


Figure 14.12 *2011 MGP Mineral Resource Estimate for combined domains by Resource Category and drill hole series, 0.36 g/t Au cut-off. Top = Plan View. Bottom = Long section looking west. (MA 2012)*

14.16 Mineral Resource Estimate Statement

The JORC categorised Mineral Resources for the Masbate Gold Project have been classified as measured, indicated and inferred confidence categories on a spatial, areal and zone basis and are listed in Table 14.16 below:

Resource Category	Tonnage	Au Grade	Contained Au
	Mt	g/t	M oz
Measured	6.4	0.77	0.16
Indicated	197.8	0.76	4.84
Total M & I	204.3	0.76	5.00
Inferred	102.8	0.86	2.83

Table 14.16 *2011 MGP Mineral Resource Estimate for combined domains, 0.36 g/t Au cut-off.*

14.16.1 Notes to accompany resource statement:

- Geological model used was sectional interpretation for 3D wireframes, veins (>0.4 g/t Au) and halo (>0.1 g/t Au) separately.
- Total of 2,149 drill holes (including 42 trenches) used, suspect and duplicate holes not used (same as 2006); represents an additional 866 holes added since 2008.
- Total of 23,934 grade control drill holes also used.
- All drill samples are composited down-hole to 2m within domains of vein (112), halo (6), dump (5), eluvial (7) or alluvial (1). Each composite only used for one domain.
- Missing samples or intervals not used, also no drill holes prior to 2000 used in the halo domain due to possible contamination at values around 0.2g/t Au.
- Top caps applied to informing samples, varied by domain and data type, halo capped at 3.5g/t.
- Total number of informing samples (composites) for veins is 27,976 drill samples and 63,314 grade control samples; for halo 35,414 drill samples and 62,388 grade control samples.
- Estimates made using Ordinary Kriging, with octants to restrict influence of grade control drilling immediately below current mine floor.
- Block size varied by resource category; 40 m(E) by 40 m(N) by 20 m(RL) for inferred, 20 m(E) by 20 m(N) by 10 m(RL) for indicated and 10 m(E) by 10 m(N) by 5 m(RL) for measured with grade control and sub-blocks for volume in all categories 5 m(E) by 5m(N) by 2.5 m(RL).
- Directional anisotropy varied by domain and sub-domain.
- Maximum search 200 m all categories.
- Inferred resource category - minimum of 3 informing samples.
- Indicated resource category – minimum of 7 informing samples and Krig Slope of 0.3.
- Measured resource category – at maximum of 32 informing samples and within 10m of grade control drilling.
- Underground voids wire framed and blocks flagged as void if over 30% of sub-block.
- Bulk Density of 2.55 t/m^3 for fresh rock, 2.40-2.55 t/m^3 for transitional and 2.37 t/m^3 for oxides.
- No mining dilution or loss, or metallurgical factors applied.
- Cut-off grade of 0.36 g/t used is similar to that currently used to define low grade mineralisation.

The information in this report that relates to Mineral Resources is based on information compiled by Mr Andrew Vigar, who is a Fellow of The Australasian Institute of Mining and Metallurgy. Mr Vigar is an employee of Mining Associates Limited and has sufficient experience which is relevant to the style of mineralisation and type of deposit under consideration and to the activity which he is undertaking to qualify as a Competent Person as defined in the 2004 Edition of the 'Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves'. Mr Vigar consents to the inclusion in this report of the matters based on this information in the form and context in which it appears.

14.17 Low Grade Stockpiles

CSA Global Pty Ltd ("CSA Global") completed a review of the low grade stockpiles constructed by FRC in April 2012. The following information is extracted from the CSA Global report (Ferguson, 2012). The dumps reviewed have been classified a JORC Indicated category due to density of grade control and industry standard survey practice. A reclassification of the dumps may occur once a reconciliation review of historical production demonstrates whether the grade control methods provide lower predicted to actual production variance.

FRC commenced gold mining on the Masbate project in May 2009 and as of the 31st October 2011, CSA Global have estimated a total of 7.4 Mt at an average gold grade of 0.55 g/t Au in the Indicated category has been stockpiled as low grade material, which would be available for processing later as deemed by the executive and technical management of the MGP (Table 14.17).

Stockpile Name	Material		Grade	Contained Au
	Type	Tonnes (kt)	g/t	koz
LG1 & LG2	Oxide	4	0.63	0.1
LG3 Main	Mixed (Oxide, Transitional / Fresh)	6,350	0.54	110.3
F12	Oxide	713	0.62	14.2
COL SB2	Oxide	255	0.57	4.7
Skyway	Oxide	15	0.61	0.3
ROM pad Sheeting	Transitional / Fresh	63	0.52	1.1
Total		**7,400**	**0.55**	**130.6**

Table 14.17 *2011 Mineral Resource Estimate, Low Grade stockpiles.*

FRC conducts specific gravity tests on its ore and waste every six months and CSA Global used the following specific gravity (SG) result in their tonnage estimations for the above low grade stockpiles tonnage estimate (Table 14.18):

Stockpile	SG
Oxide	1.9
Transition & Fresh	2.2

Table 14.18 *Low Grade stockpile specific gravity values.*

Stockpile tonnages are estimated using an 80% swell factor and checked monthly through tonnage reconciliations. FRC management states the reconciliations to be within 5% for tonnage from mine to mill. CSA Global has not been able to review reconciliations of tonnage and grades, however from their observations during a visit on 25 April 2012, CSA Global believes the density estimations to be appropriate.

The Masbate low grade stockpiles have been assigned a JORC Indicated resource category based upon geological knowledge. As the dumps have been sampled through grade control, and their volume determined by standard industry practice survey, an Indicated category is deemed appropriate. A future study on the historical reconciliation of the ore and variance from grade control to actual mined and milled may result in a higher grade confidence and a reclassification to JORC Measured category (Ferguson 2012).

14.18 Discussion

This current mineral resource estimate is an update to the May 2008 work having incorporated new drilled results conducted since 2008 plus reconciliation with the mine production up to October 2011. The current model shows a good correlation over this period between estimated and grade control results, supporting the drop in overall grade noted both between the 2008 to 2011 estimates and between 2008 and the grades delivered to the processing plant.

15 MINERAL RESERVE ESTIMATES

15.1 2011 Mineral Reserve

The mining studies were undertaken by Golder Associates and are based on the 2011 Mineral Resources as discussed in Section 14. Within the mineral reserve estimate section all costs and revenues are expressed in United States dollars (USD). A Whittle based optimisation based was undertaken using a USD 1,300 / oz base gold price. At the time of the optimisation the spot price was in excess of USD 1,600 / oz. Final pit designs were completed by Golder Associates utilising the recommended slope parameters and considering the previous underground mining and the location of geological faults.

The assessment of Mineral Reserves is based on the material classified as Proven and Probable in the resource assessment which is contained within the final pit design and which exceeds the nominal cut-off grades as shown in Table 15.1.

Material Type	Fresh	Transition	Oxide
Minimum cut-off Grade (Au g/t)	0.43	0.39	0.36

Table 15.1 *Nominal cut-off grade by material type, 2011 MGP mineral reserve estimate.*

Table 15.2 is a summary of the updated mineral reserves estimates of October 2011 for Masbate.

Category	Material Type	Tonnage (Mt)	Gold grade (g/t)	Contained Au (000's oz)
Proven	Oxide	2.8	0.66	58
	Transition	1.0	0.74	25
	Fresh	1.9	0.89	56
	Total	**5.7**	**0.75**	**139**
Probable	Oxide	20.5	0.68	445
	Transition	34.5	0.79	882
	Fresh	50.5	0.97	1,572
	Low grade Stockpiles	7.4	0.55	131
	Total	**112.9**	**0.83**	**3,029**

Table 15.2 *October 31st 2011 MGP mineral reserve estimate.*

The mineral reserve is diluted by 5 % assuming a gold grade of 0.17 g/t Au and had a mining recovery 97.5 % applied. The mineral reserve is contained within five pits. The estimated waste within the pits is 255 Mt at a strip ratio of 2.3:1 t:t.

Contained within the final pit designs is about 12 Mt at 1.0 g/t of Inferred material.

Figures 15.1 to Figure 15.9 show the final open pit designs for the various areas in both plan and cross section views.

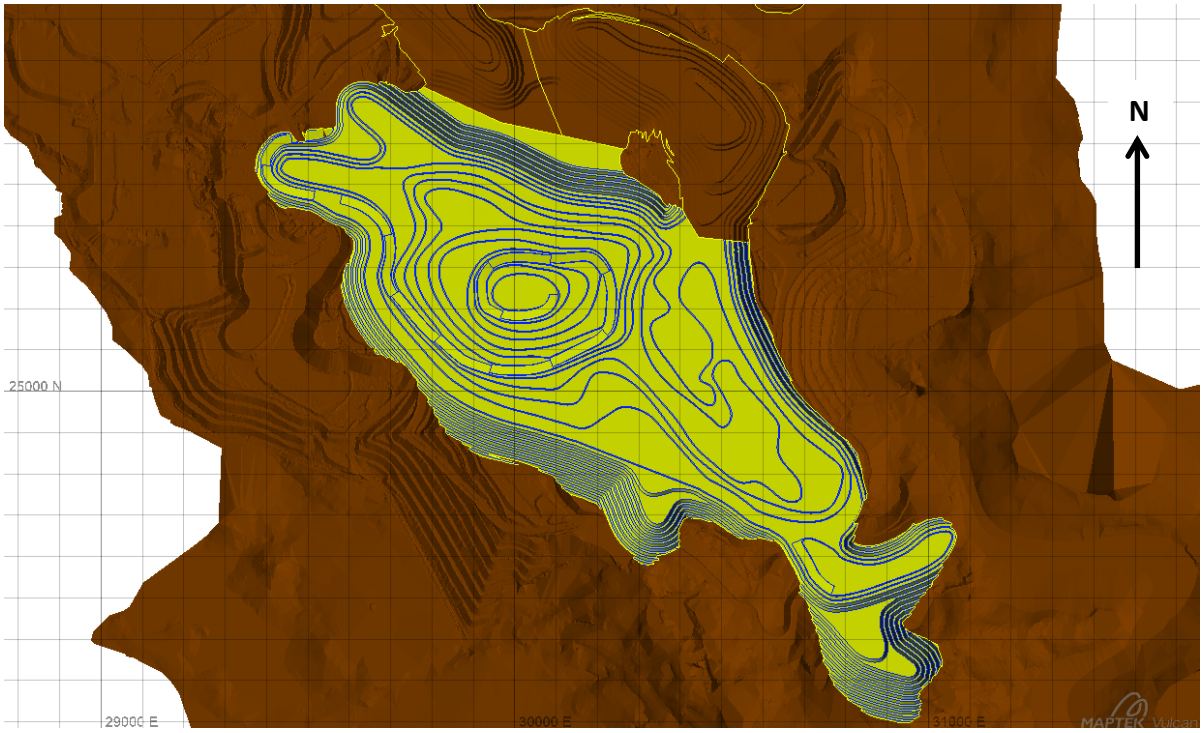


Figure 15.1 *Main Vein pit design, plan view. 2011 MGP mineral reserve estimate. (Golder 2012)*

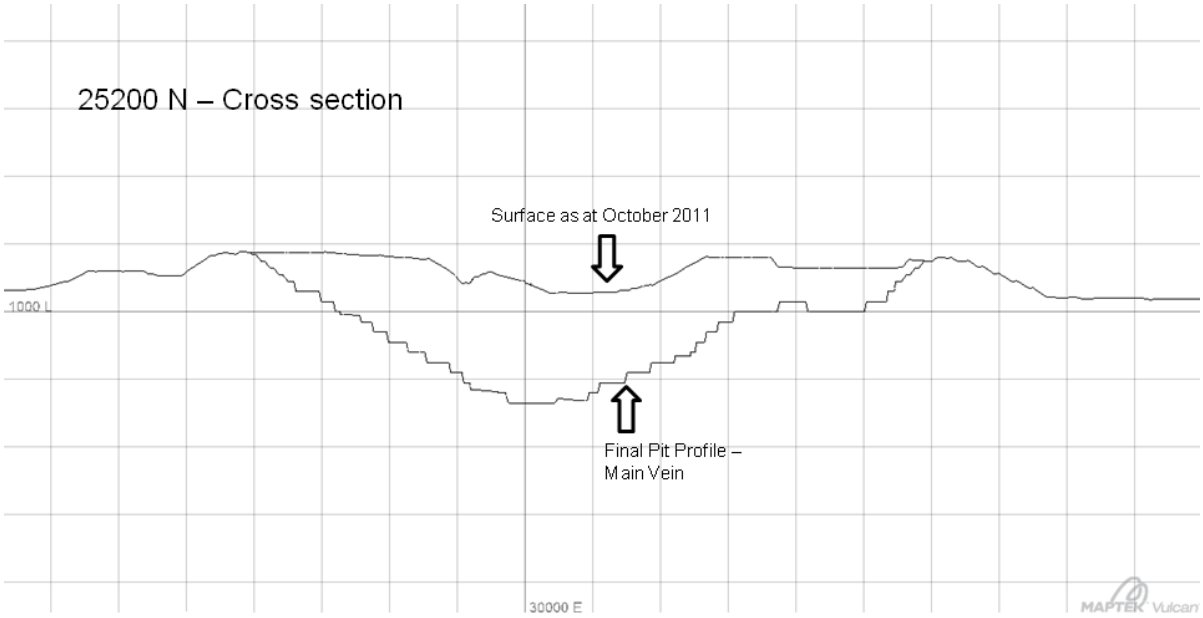


Figure 15.2 *Main Vein pit design, cross section through 25,200 N (looking north). 2011 MGP mineral reserve estimate.*

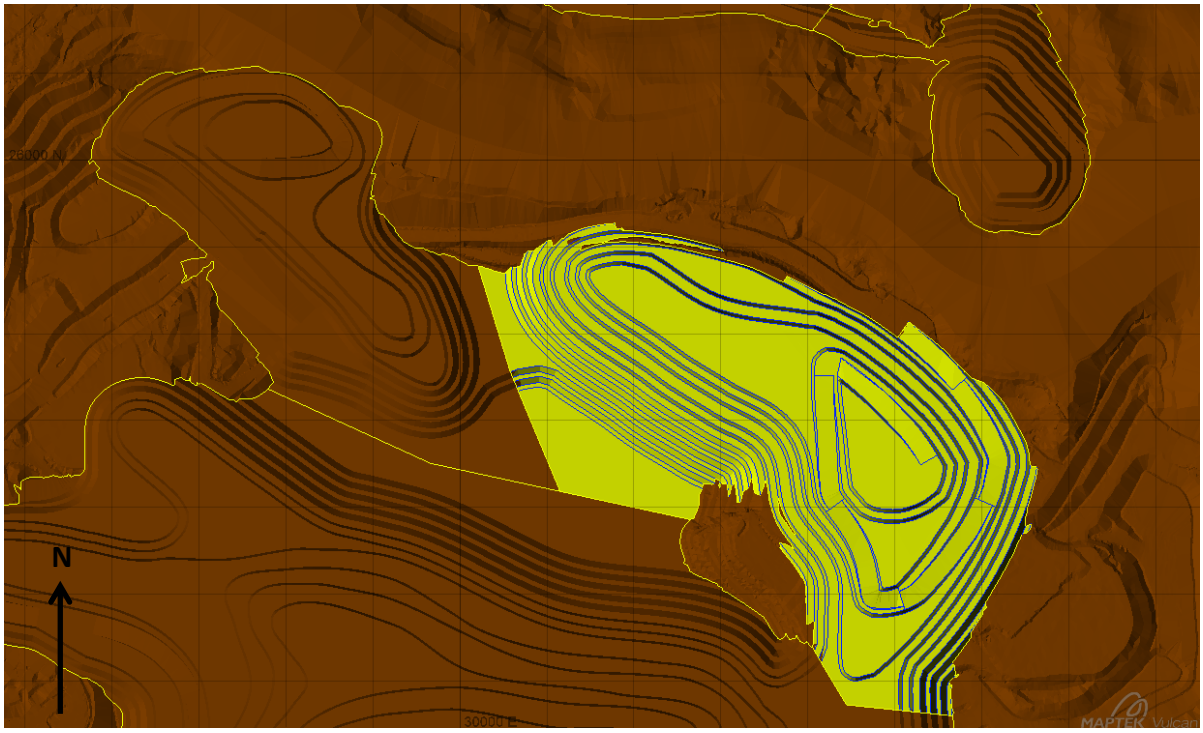


Figure 15.3 *Holy Moses – Basalt East pit design, plan view. 2011 MGP mineral reserve estimate. (Golder 2012)*

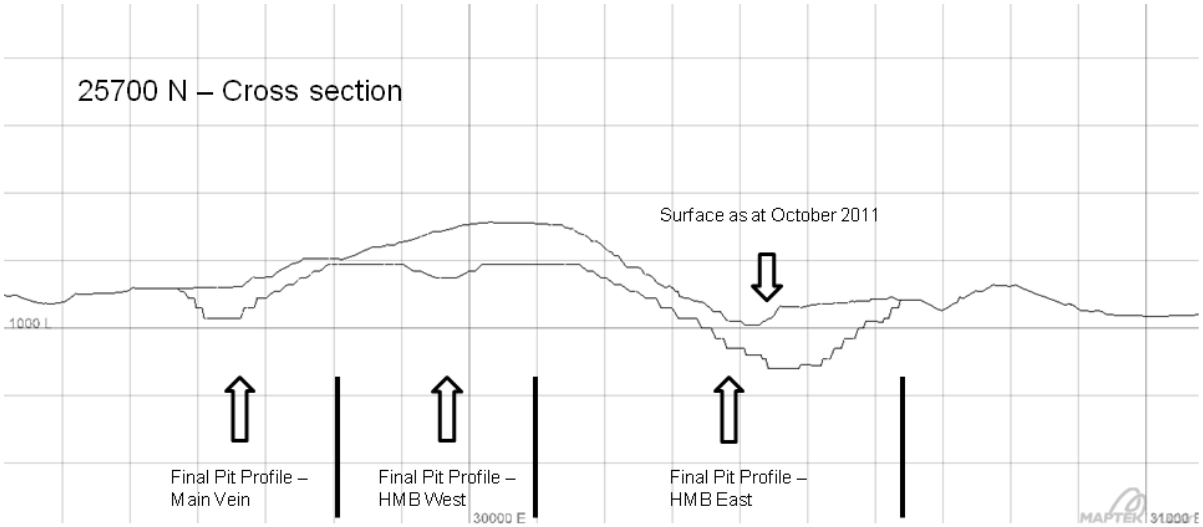

25700 N – Cross section
Surface as at October 2011
1000 L
Final Pit Profile – Main Vein
Final Pit Profile – HMB West
Final Pit Profile – HMB East
30000 E

Figure 15.4 *Holy Moses – Basalt East pit design, cross section through 25,700 N (looking north). 2011 MGP mineral reserve estimate.*

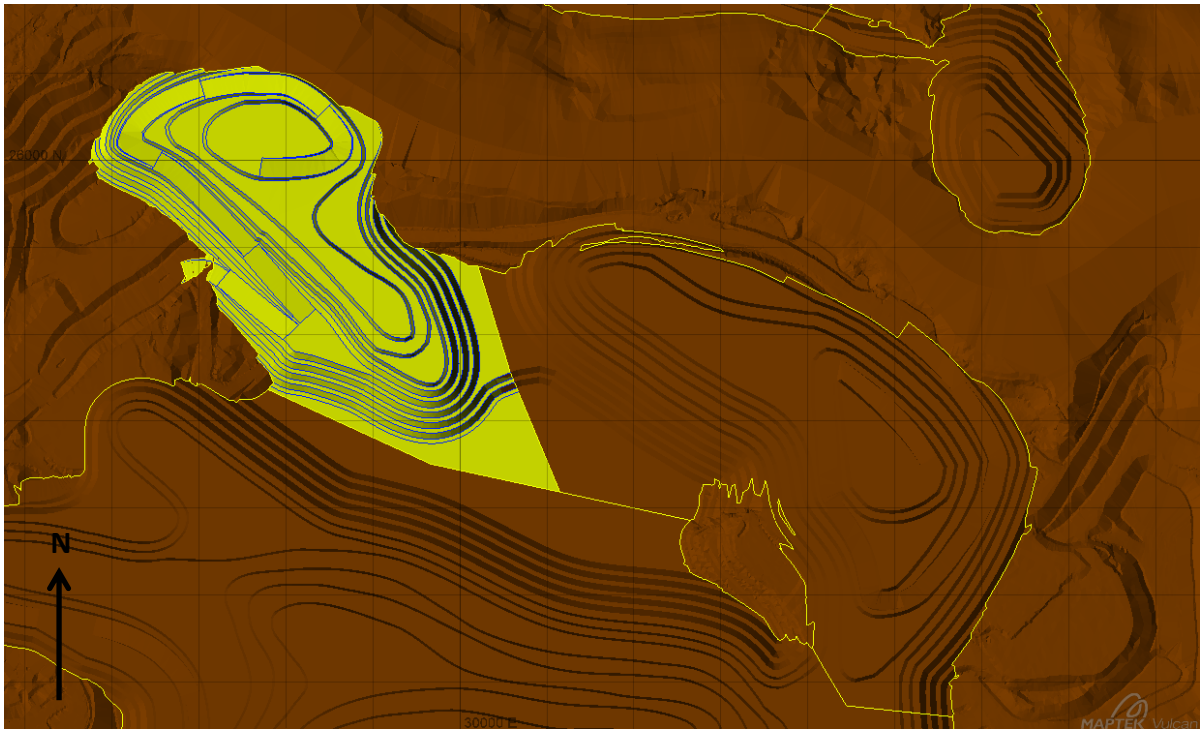


Figure 15.5 *Holy Moses – Basalt West pit design, plan view. 2011 MGP mineral reserve estimate. (Golder 2012)*

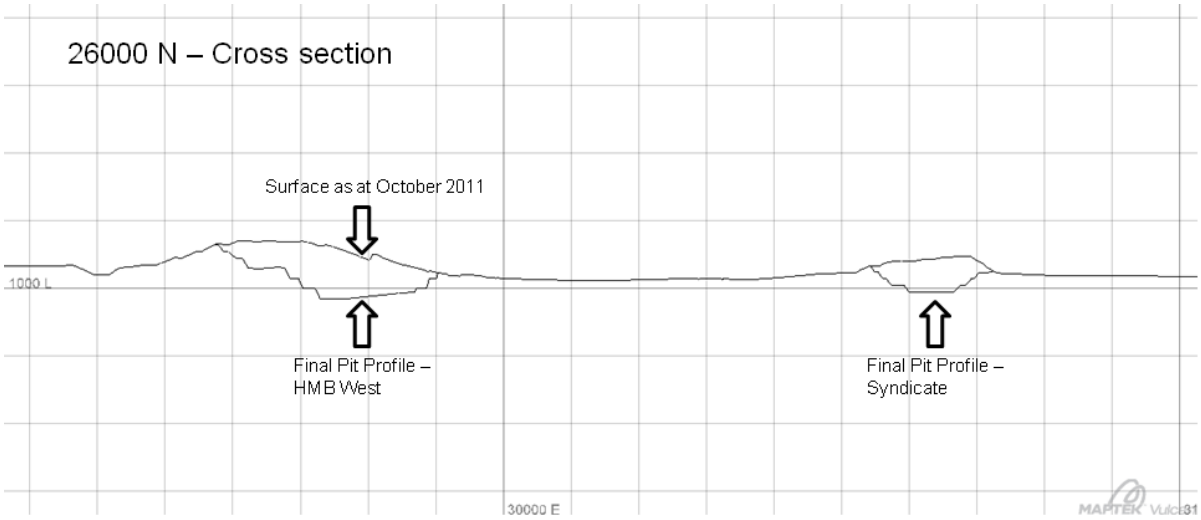


Figure 15.6 *Holy Moses – Basalt West and Syndicate pit designs, cross section through 26,000 N (looking north). 2011 MGP mineral reserve estimate.*

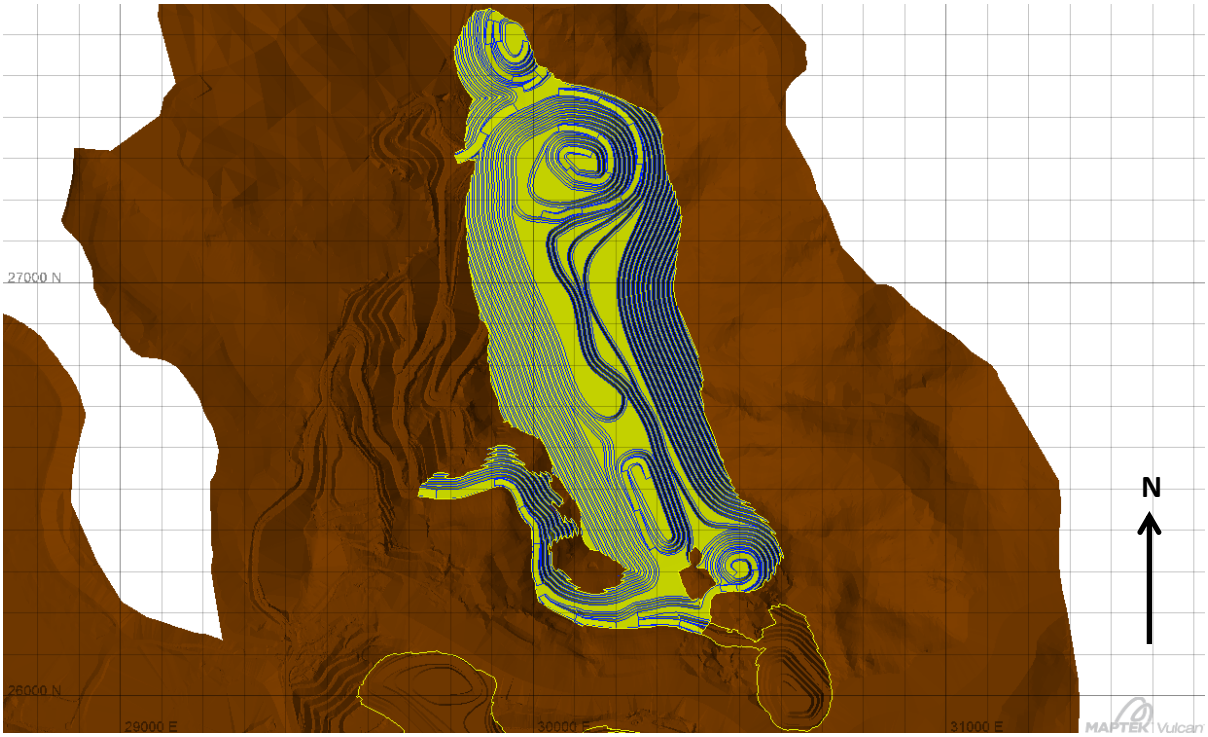


Figure 15.7 *Colorado pit design, plan view. 2011 MGP mineral reserve estimate. (Golder 2012)*

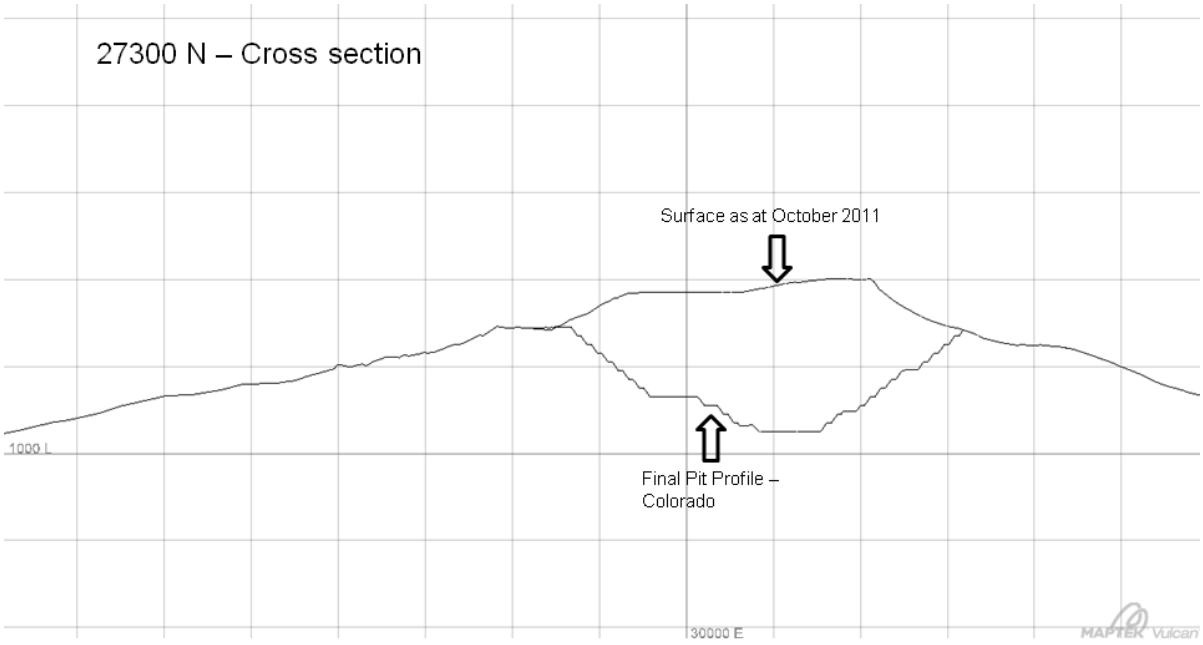


Figure 15.8 *Colorado pit design, cross section through 27,300 N (looking north). 2011 MGP mineral reserve estimate.*

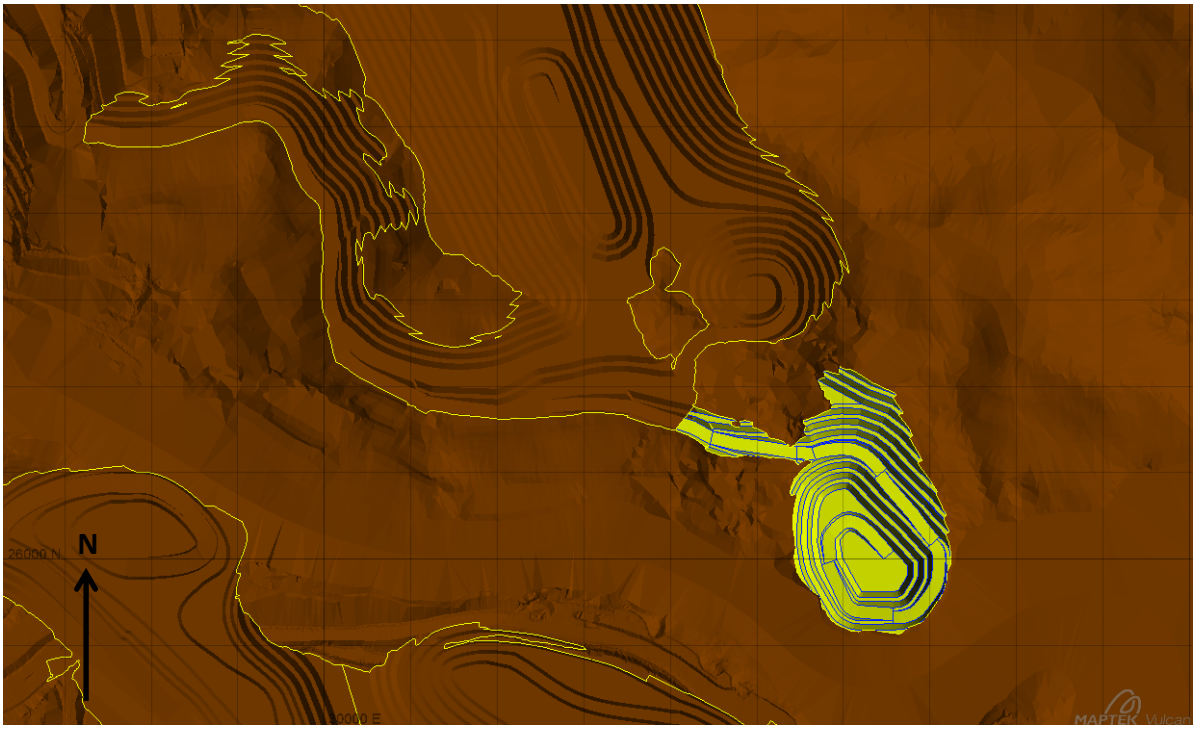


Figure 15.9 *Syndicate pit design, plan view. 2011 MGP mineral reserve estimate. (Golder 2012)*

15.2 Key Parameters used in the Reserve Estimate

The cut-off grade calculations used a gold price of USD 1,300 / oz, a metallurgical recovery based on ore type, and the operating costs based upon the budget for the existing mine schedule over the next three years, averaged for each mining area. The processing costs are based upon the budget for 6.5 million tonnes per annum.

The cut-off grade for this reserve has been derived from the following equation:

$$COG = \frac{CostP}{(P)(MillR)}$$

Where: COG = Cut-off grade (g/t)
CostP = Processing costs (USD/t ore)
P = Net price of metal (USD/g)
MillR = Mill recovery (proportion)

The net price was calculated by converting USD / oz into USD / g and subtracting a sales cost of 2 % and a royalty of 2 %. The sales cost was applied for the export tax and a royalty for the government tax.

The open pit ore reserve is based on a cut-off grade for each of the rock types; that is, 0.36 g/t Au for oxide material, 0.39 g/t Au for transitional material and 0.43 g/t Au for fresh material.

Material type	Fresh	Transition	Oxide
Gross Price (USD/g)	41.80	41.80	41.80
Royalty (USD/g)	0.84	0.84	0.84
Selling cost (USD/g)	0.84	0.84	0.84
Net Price (USD/g)	40.12	40.12	40.12
Nominal processing cost (USD/t)	12.82	12.81	12.81
Overall Recovery (%)	74.0	82.0	88.0
Cut-off Grade (g/t)	**0.43**	**0.39**	**0.36**

Table 15.3 *Cut-off grade calculation, 2011 MGP mineral reserve estimate.*

Mining dilution of 5 % at 0.17 g/t grade and mining recovery of 97.5 % have also been applied to the reserve.

Whittle software was used to optimize the final pit shell, and the detailed design was done in Vulcan Mine Planning software.

Mineral reserves estimated for this deposit are located within the physical boundaries of the MGP and are not materially affected by any known mining, metallurgical, infrastructure, permitting or other relevant factors.

15.3 Open Pit Optimisation

The pit optimization was achieved using Whittle software. Golder has applied slope constraints, mining costs, processing costs and metallurgical recoveries to the re-blocked geological block model.

The initial cost parameters were provided by FRC and were modified by Golder to include mining administrator cost that were not consistently included in the ore and the waste costs across all areas.

15.3.1 Metal Price

The base metal price supplied by FRC was USD 1,300/oz. of gold.

15.3.2 Royalties

The royalty rate supplied by FRC is 2.0 % of recovered gold.

15.3.3 Dilution and Ore Loss

Mining dilution of 5% at 0.00 g/t grade and 5% ore loss were applied (in Whittle) though for scheduling and reserve estimation a dilution grade of 0.17 g/t Au was used with a 2.5% mining loss. This was based on a more detailed review of the low grade areas surrounding the above cut-off material.

15.3.4 Plant Throughput

Plant throughput supplied by FRC was 6.5 Mt per year.

15.3.5 Mining Limits

Mining fleet capacity limits were not used.

15.3.6 Physical Boundaries

There were no limitations applied to the optimisation. That is, the FRC lease, mineral rights boundaries or the influence of the Guinobatan River did not constrain the optimisation shells.

15.3.7 Discount Rate

A discount rate of 10% was applied in determining Present Values for optimised pits.

15.3.8 Capital Costs

No capital costs were applied to the optimisation.

15.3.9 Slope Angles

An overall slope angle for each region has been used for the optimisation (Table 15.4).

Wall	Overall Angle
All Walls (default)	43°
Colorado Oxide Material	40°
Sector 4 (Main Vein Area)	35°

Table 15.4 *Overall slope angles, open pit optimisation, 2011 MGP mineral reserve estimate.*

Figure 15.10 shows the location of each of the regions.

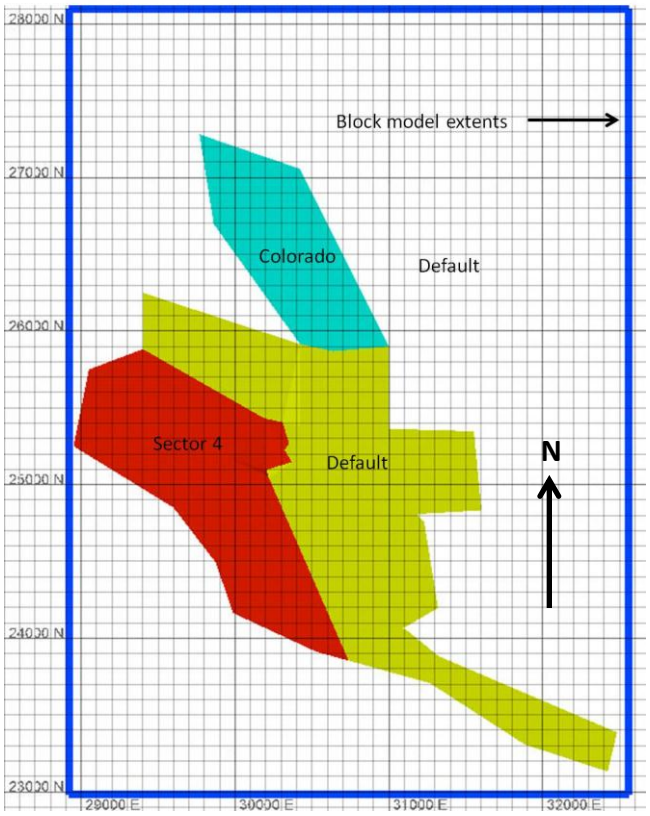


Figure 15.10 *Location of pit wall angle sectors, 2011 MGP mineral reserve estimate. (Golder 2012)*

The basis of the location of each of the regions is from George, Orr and Associates recommendations in a report dated 17 November 2009 (Figure 15.11).

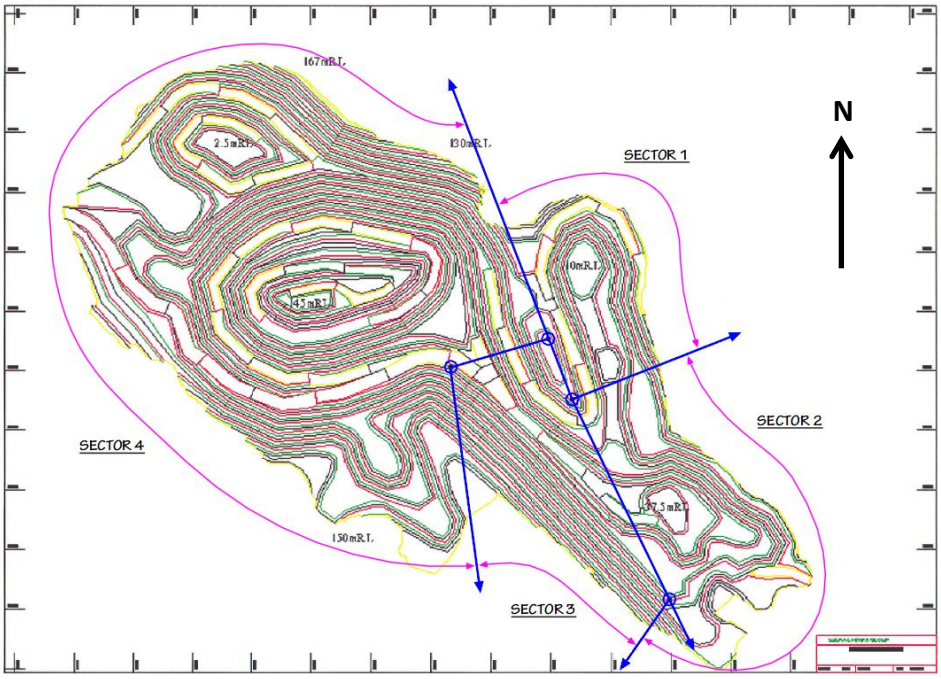


Figure 15.11 *Main Vein geotechnical sectors. (George Orr & Associates 2009)*

Sector 4 compared to Intermine Engineering Consultants design aligns with the George, Orr and Associates recommendations (Figure 15.12).

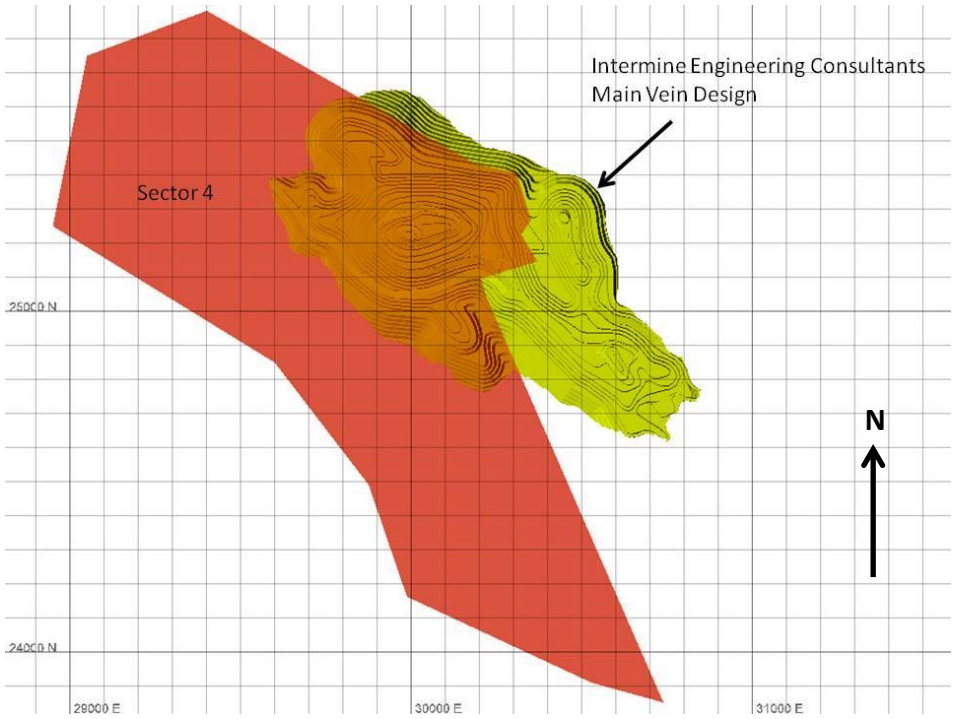


Figure 15.12 *Intermine Engineering Consultants Main Vein pit design and location of Sector 4wall angle, 2011 MGP mineral reserve estimate. (Golder 2012)*

15.3.10 Process Recovery

Table 15.5 summarises the assumed process recoveries by element used in the pit optimisation.

Type	Oxide (%)	Transitional (%)	Fresh (%)
High grade ore (≥1.0 g/t Au)	93	86	82
Medium grade ore (≥0.7 and <1.0 g/t Au)	88	82	74
Low grade ore (<0.7 g/t Au)	88	82	74

Table 15.5 *Process recoveries, open pit optimisation, 2011 MGP mineral reserve estimate.*

15.3.11 Mining & Processing Costs

The mining and processing costs were assigned by region as shown in Figure 15.13.

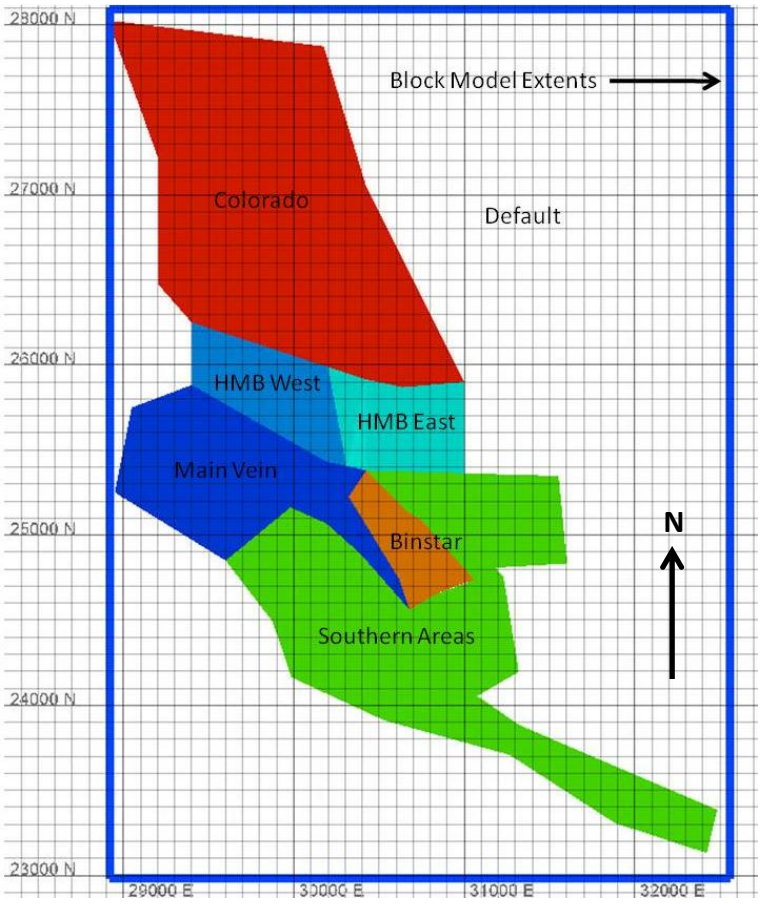


Figure 15.13 *Mining and Process cost regions, 2011 MGP mineral reserve estimate. (Golder 2012)*

The mining costs were extracted from the budget for the next three years. The operating mining contractor has a margin of up to 16% over the mining costs (except fuel).

The fixed costs were converted into a cost per bank cubic metre (BCM) by dividing the cost by the total BCM moved (Table 15.6).

Fixed Cost Item	Cost (USD/BCM)
Ancillary Equip Support	0.15
Additional labour	0.42
Equip Fixed costs	0.60*
Dewatering	0.02
Overheads	0.22
Drill & Blast	1.27

Table 15.6 *Fixed costs (pre-margin), 2011 MGP mineral reserve estimate.*

The load and haul cost was averaged per level for each stage. The average cost per BCM for each pit for load and haul was also calculated (Table 15.7).

Load and Haul Area	Cost (USD/BCM)
Main Vein Area (MVA)	0.78
Binstar	0.46
HMB East	0.78
HMB West Area	0.70
Colorado Area	0.84
Syndicate Area	0.78

Table 15.7 *Load and Haul costs (pre-margin), 2011 MGP mineral reserve estimate.*

The fuel used to operate the mine does not have a margin applied (Table 15.8).

Item	Cost (USD/BCM)
Fuel	1.53

Table 15.8 *Fuel cost, 2011 MGP mineral reserve estimate.*

The grade control cost was assigned to the ore mining cost only, hence used the total ore volume moved. Mine administration costs however, are applied to both the ore and waste, therefore used the total volume moved. FRC is responsible for both the grade control and the mine administration costs hence a margin has not been applied to these costs (Table 15.9).

Item	Cost (USD/BCM)
Grade Control	0.46
Mine Administration	0.50

Table 15.9 *Mining costs (accountable to FRC), 2011 MGP mineral reserve estimate.*

The drill and blast costs were allocated by material types (Table 15.10).

Material type	Drill and blast cost (USD/BCM)
Oxide	0.85
Transition	1.09
Fresh	1.41

Table 15.10 *Drill and Blast costs, 2011 MGP mineral reserve estimate.*

The mining costs associated with ore and waste were estimated on a cost per tonne basis so that they can be used in the optimisation process. The densities for oxide, transition and fresh material used to convert the cost per BCM to a cost per tonne are shown in Table 15.11.

Material type	Density (t/BCM)
Oxide	2.37
Transition	2.40
Fresh	2.50

Table 15.11 *Material density values, 2011 MGP mineral reserve estimate.*

The cost structures were applied to each different area. A typical structure is shown for Main Vein, the Southern Areas and the default region in Table 15.12.

Type	Mining costs (USD/t)			Processing costs (USD/t)		
	Oxide	Transition	Fresh	Oxide	Transition	Fresh
min	1.96	2.05	2.12	12.81	12.81	12.82
max	2.58	2.66	2.70	13.51	13.51	13.49

Table 15.12 *Main Vein, Southern Areas and default region cost structure summary, 2011 MGP mineral reserve estimate.*

15.3.12 Resource Model

The Masbate resource model provided by FRC was a Surpac block model titled "masbate_ma_master20120108.mdl". Table 15.13 summarises the combined Mineral Inventory for the Masbate deposit at 0.00 and 0.36 g/t Au.

Resource Category	Tonnage (Mt)	Contained Au (Moz)	Grade (g/t)	Tonnage (Mt)	Contained Au (Moz)	Grade (g/t)
Cut-off grade (g/t)	0.00			0.36		
Measured	14	0.21	0.46	6	0.16	0.77
Indicated	1,481	10.76	0.23	198	4.84	0.76
Total Measured and Indicated	**1,496**	**10.97**	**0.23**	**204**	**5.00**	**0.76**
Inferred	346	3.93	0.35	103	2.83	0.86
Total	**1,841**	**14.90**	**0.25**	**307**	**7.82**	**0.79**

Table 15.13 *Mineral Inventory, 2011 MGP mineral reserve estimate.*

For the purposes of the optimisation Measured and Indicated Resources were consider to be part of the combined Mineral Inventory. The Inferred Resource was not assigned any value and was considered waste.

The model has the following major model parameters (Table 15.14):

Characteristic	X (Easting)	Y (Northing)	Z (RL)
Parent Block size (m)	40	40	20
Minimum Sub Block Size (m)	5	5	2.5
Model origin (m)	28,920	22,980	-400
Extent (m)	3,640	5,120	760
Number of parent blocks	91	128	38

Table 15.14 *2011 Resource Model Parameters 2011 MGP mineral reserve estimate.*

The original model was reblocked in Vulcan to create a regularised model 10 m X x 10 m Y x 5 m Z. This estimated and accounted for the smallest mining unit (SMU) of the operation.

Table 15.15 summarises the mineral inventory by resource category within the re-blocked model.

Resource Category	Tonnage (Mt)	Contained Au (Moz)	Grade (g/t)	Tonnage (Mt)	Contained Au (Moz)	Grade (g/t)
Cut-off grade (g/t)	0.00			0.36		
Measured	14	0.20	0.46	7	0.16	0.73
Indicated	1,455	10.75	0.23	225	5.00	0.69
Total Measured and Indicated	**1,469**	**10.95**	**0.23**	**232**	**5.15**	**0.69**
Inferred	349	3.67	0.33	107	2.56	0.74
Total	**1,818**	**14.63**	**0.25**	**339**	**7.71**	**0.71**

Table 15.15 *Mineral Inventory for 10 m x 10 m x 5 m block model, 2011 MGP mineral reserve estimate.*

In this case, regularising the block model has resulted in a 10% drop in grade and a 10% increase in tonnage with a small drop in overall contained metal.

This model was subsequently exported from Vulcan into Whittle format.

15.3.13 Optimisation Results

The revenue factor of 1.23 was chosen for the pit shell using the price of USD 1,300/oz for cut-off grade purposes to classify ore and waste. The revenue factor of 1.23 or pit 8 was chosen because the spot price for gold was over USD 1,600/oz during October 2011. Pit 8 results in 110.0 Mt of ore at 0.82 g/t gold at a strip ratio of 2.29 t:t (Table 15.16).

Pit number 8 produces two major areas of mining in Colorado in the north and Main Vein, and the associated pits of Holy Moses Basalt (HMB) East and West (Figure 15.14).

Pit no.	Revenue factor	Ore (Mt)	Gold Grade (g/t)	Recovered Gold (000's oz)	Waste (Mt)	Rock (Mt)	Strip ratio (t:t)
1	0.69	34.2	0.95	875	48.4	82.6	1.41
2	0.77	54.1	0.92	1,319	89.3	143.5	1.65
3	0.85	68.7	0.90	1,625	122.4	191.0	1.78
4	0.92	77.2	0.88	1,783	140.2	217.4	1.82
5	1.00	86.3	0.86	1,946	161.7	248.0	1.87
6	1.08	94.9	0.85	2,103	189.8	284.7	2.00
7	1.15	101.2	0.83	2,210	210.4	311.7	2.08
8	1.23	110.9	0.82	2,381	254.2	365.1	2.29
9	1.31	116.6	0.82	2,479	283.2	399.7	2.43
10	1.38	121.5	0.81	2,565	313.5	435.0	2.58
11	1.46	125.4	0.81	2,628	338.0	463.4	2.70
12	1.54	130.3	0.80	2,714	379.0	509.2	2.91
13	1.62	133.0	0.80	2,759	403.5	536.6	3.03
14	1.69	135.8	0.80	2,802	431.1	566.9	3.18
15	1.77	138.8	0.79	2,852	467.7	606.6	3.37
16	1.85	141.5	0.79	2,892	501.2	642.7	3.54
17	1.92	143.1	0.79	2,916	525.3	668.3	3.67

Table 15.16 *Overall optimization results, 2011 MGP mineral reserve estimate.*

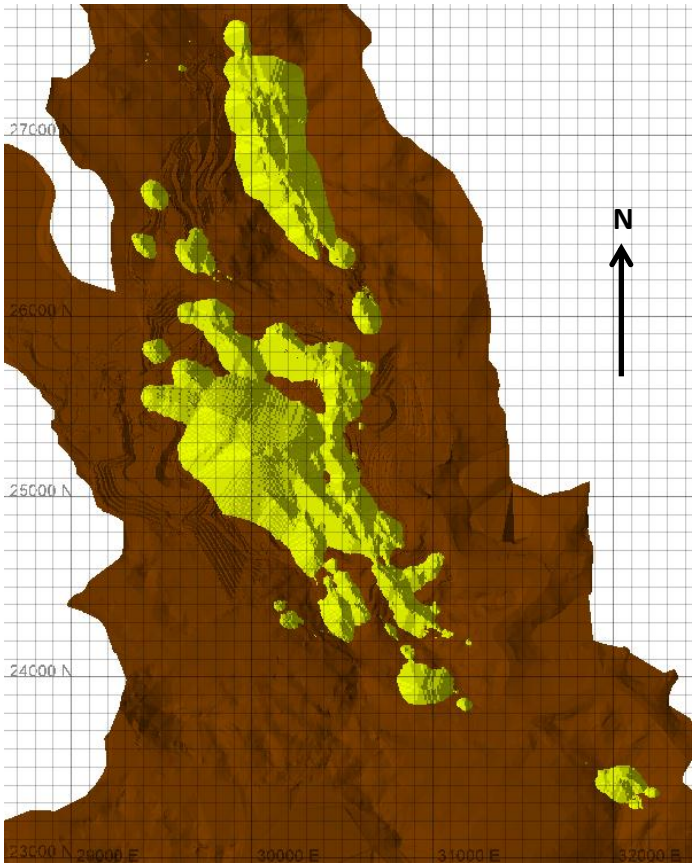


Figure 15.14 *Whittle Optimisation pit number 8, 2011 MGP mineral reserve estimate. (Golder 2012)*

15.4 Pit Design Methodology

15.4.1 General

Following discussions with FRC, Pit 8 was selected as the basis for the detailed pit design as this modelled the price and cost structure for the Masbate Project going forward.

Mine designs were completed for each of the following deposits (Figure 15.15).

- Main Vein
- Colorado
- HMB East
- HMB West
- Syndicate

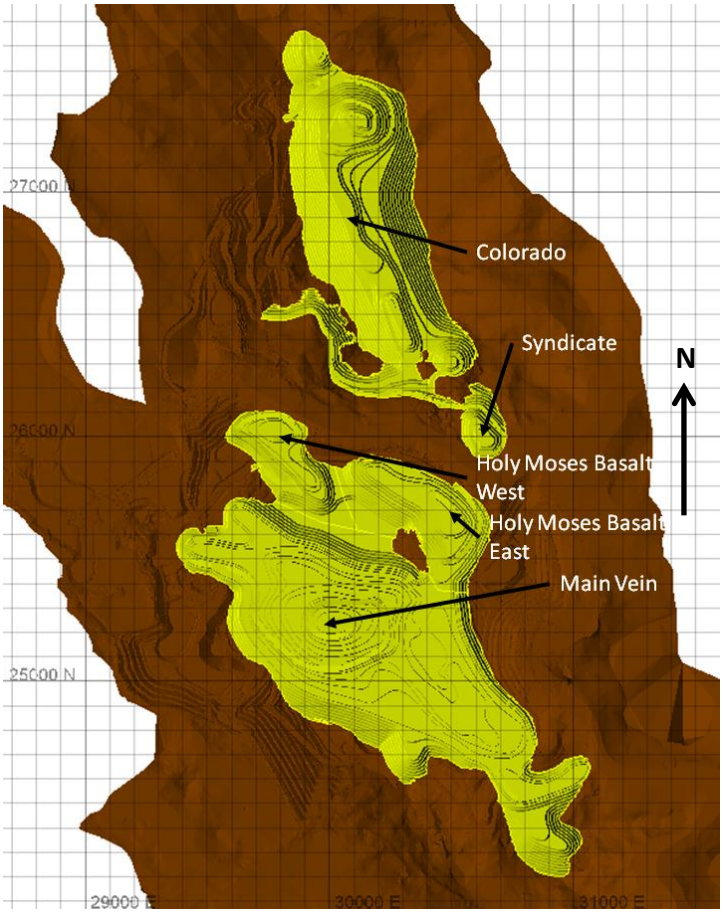


Figure 15.15 *Life of Mine pit designs, 2011 MGP mineral reserve estimate. (Golder 2012)*

The two surfaces used to estimate the following mineral reserves was provided by FRC, that is the end of month (for October 2011) topography (ma_eom_all1011) and the pre-mining surface (ma_topo_premine20111006).

15.4.2 Main Vein Design

Pit design parameters for the Main Vein pit included:

- a double lane ramp width of 22 m at a gradient of 1 in 10;
- in fresh rock (below 1045 m RL) 15 m high batters at 80 degrees with 8 m berm widths; and
- in weathered material (above 1045 m RL) 10 m high batters at 55 degrees and 5 m berms.

The Main Vein design contained 1.8 Mt at 0.74 g/t Au of proven reserve and 56.3 Mt at 0.91 g/t Au of probable reserve (Figure 15.17).

Category	Material type	Tonnage (Mt)	Gold grade (g/t)	Contained Au (000's oz)
Proven	Oxide	0.6	0.68	12
	Transition	0.5	0.72	12
	Fresh	0.7	0.81	17
	Total	**1.8**	**0.74**	**42**
Probable	Oxide	7.0	0.61	136
	Transition	4.8	0.78	121
	Fresh	44.5	0.97	1,391
	Total	**56.3**	**0.91**	**1,648**

Table 15.17 *Main Vein Reserve summary, 2011 MGP mineral reserve estimate.*

The estimated waste within the Main Vein pit is 143 Mt at a strip ratio of 2.5:1 t:t.

Contained within the Main Vein design is about 6.2 Mt at 1.0 g/t of Inferred material.

An isometric view of the Main Vein pit can be seen in Figure 15.16.

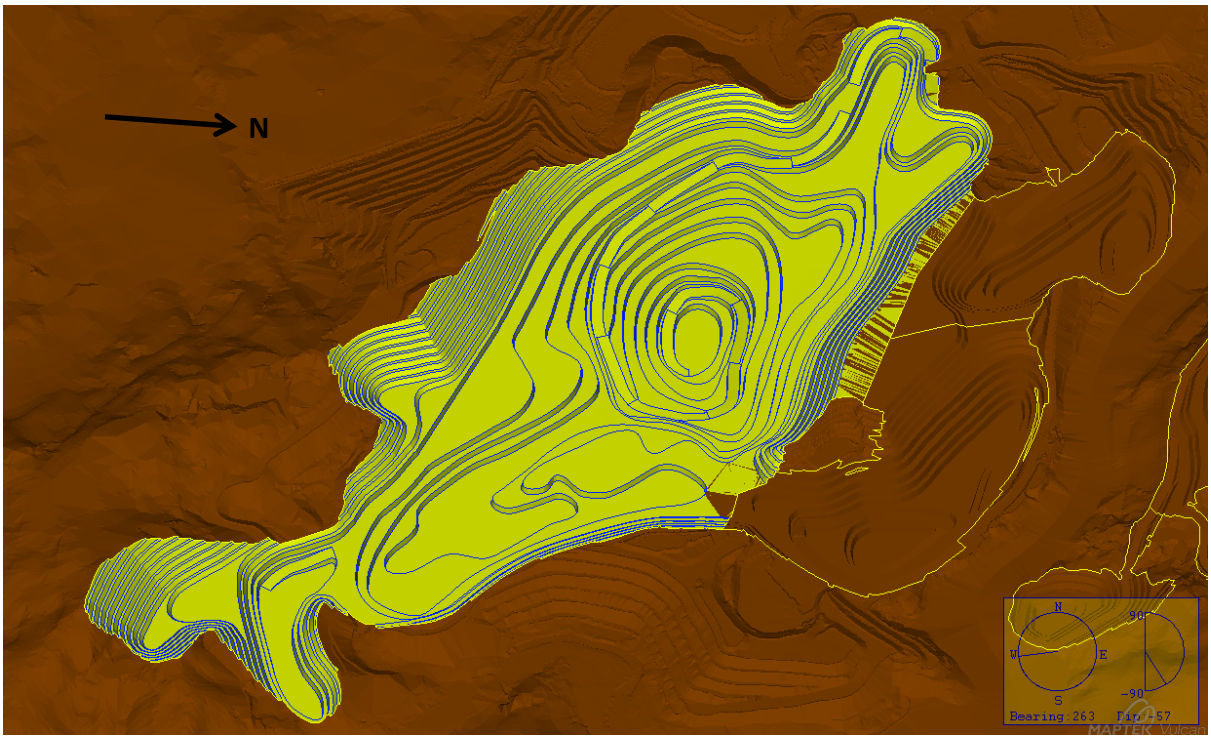


Figure 15.16 *Main Vein pit design (view = bearing 263°, dip -57°), 2011 MGP mineral reserve estimate.(Golder 2012)*

15.4.3 Colorado Design

Design parameters for the Colorado pit included:

- in the northern section of the Colorado design (approximately 26,650 m N to 27,700 m N) a double lane ramp width of 22 m at a gradient of 1 in 10 from 145 m RL to 125 m RL due to traffic considerations;
- in the northern section of the Colorado design (approximately 26,650 m N to 27,700 m N) a single lane width of 15 m at a gradient of 1 in 10 from 125 m RL to 25 m RL (lowest level);
- in the southern section of the Colorado design (approximately 25,900 m N to 26,650 m N) a single lane width of 15 m at a gradient of 1 in 10;
- in weathered material 10 m high batters at 55 degrees and 5 m berms, and
- in fresh material 15 m high batters at 80 degrees and 8 m berms.

The Colorado design contains 2.6 Mt at 0.67 g/t of proven reserve and 42.5 Mt at 0.77 g/t of probable material (Table 15.18).

Category	Material type	Tonnage (Mt)	Gold grade (g/t)	Contained Au (000's oz)
Proven	Oxide	2.2	0.66	46
	Transition	0.4	0.77	11
	Fresh	0.0	0.00	0
	Total	**2.6**	**0.67**	**57**
Probable	Oxide	13.0	0.71	296
	Transition	29.2	0.79	746
	Fresh	0.4	0.62	8
	Total	**42.5**	**0.77**	**1,049**

Table 15.18 *Colorado Reserve summary, 2011 MGP mineral reserve estimate.*

The estimated waste within the Colorado pit is 85 Mt at a strip ratio of 1.9:1 t:t.

Contained within the Colorado design is about 5.3 Mt at 1.0 g/t of Inferred material.

An isometric view of the Colorado design can be seen in Figure 15.17.

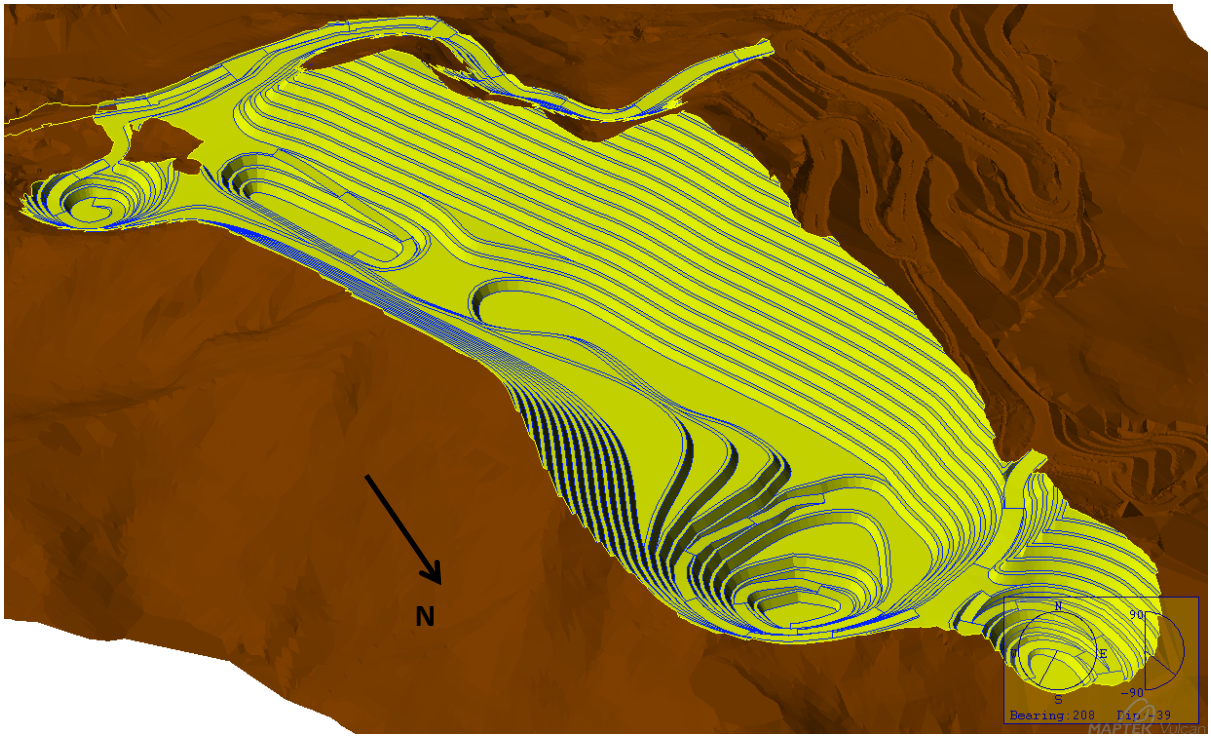


Figure 15.17 *Colorado pit design (view = bearing 208°, dip -39°), 2011 MGP mineral reserve estimate. (Golder 2012)*

15.4.4 HMB East Design

The pit design parameters for HMB East pit included:

- a double lane ramp width of 22 m at a gradient of 1 in 10;
- in fresh rock 15 m high batters at 75 degrees with 8 m berm widths;
- in weathered material 10 m high batters at 55 degrees and 5 m berms; and
- a 12 m wide berm on the western wall was incorporated at 15 m RL and a 12 m wide berm on the eastern wall was placed at -15 m RL, to assist in catching any potential rock fall.

Inside the HMB East design there is 1.4 Mt at 0.92 g/t Au of proven material and 4.9 Mt at 0.82 g/t Au of probable reserve (Table 15.19).

Category	Material type	Tonnage (Mt)	Gold grade (g/t)	Contained Au (000's oz)
Proven	Oxide	0.0	0.58	0
	Transition	0.1	0.74	2
	Fresh	1.3	0.93	38
	Total	**1.4**	**0.92**	**40**
Probable	Oxide	0.3	0.67	7
	Transition	0.3	0.66	6
	Fresh	4.2	0.84	115
	Total	**4.9**	**0.82**	**128**

Table 15.19 *Holy Moses – Basalt (HMB) East Reserve summary, 2011 MGP mineral reserve estimate.*

The estimated waste within the HMB East pit is 12 Mt at a strip ratio of 1.9:1 t:t.

Contained within the HMB East design is about 0.2 Mt at 1.0 g/t of Inferred material.

The HMB East design can be seen in an isometric view in Figure 15.18.

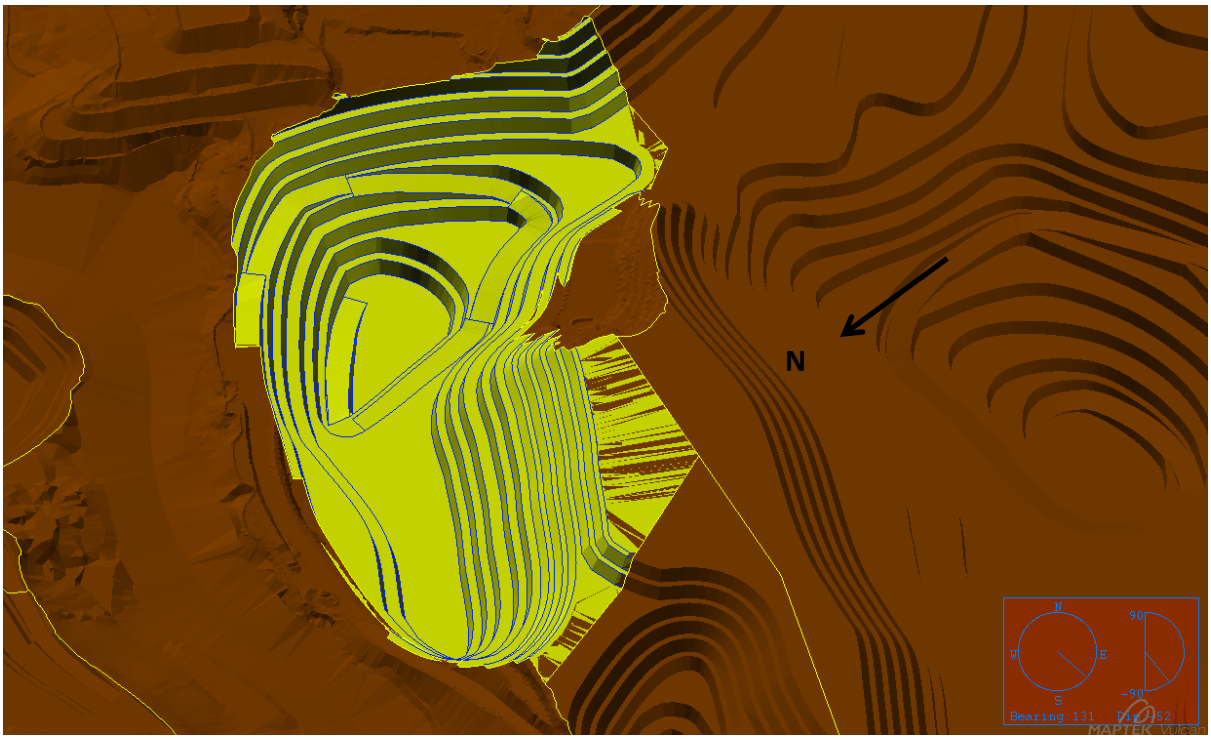


Figure 15.18 *Holy Moses – Basalt East pit design (view = bearing 131°, dip -52°), 2011 MGP mineral reserve estimate. (Golder 2012)*

15.4.5 HMB West Design

The pit design parameters for HMB West included:

- a double lane ramp width of 22 m at a gradient of 1 in 10;
- in fresh rock 15 m high batters at 75 degrees with 8 m berm widths; and

- in weathered material 10 m high batters at 55 degrees and 5 m berms.

The HMB West design encases less than 0.1 Mt at 1.95 g/t Au of proven material and 1.2 Mt at 1.40 g/t Au of probable reserve (Table 15.20).

Category	Material type	Tonnage (Mt)	Gold grade (g/t)	Contained Au (000's oz)
Proven	Oxide	0.0	0.00	0
	Transition	0.0	0.00	0
	Fresh	<0.1	1.95	<1
	Total	**<0.1**	**1.95**	**<1**
Probable	Oxide	0.1	0.99	2
	Transition	0.1	1.30	3
	Fresh	1.1	1.43	49
	Total	**1.2**	**1.40**	**54**

Table 15.20 *Holy Moses – Basalt (HMB) West Reserve summary, 2011 MGP mineral reserve estimate.*

The estimated waste within the HMB West pit is 14 Mt at a strip ratio of 11.4:1 t:t.

Contained within the HMB West design is about 0.3 Mt at 1.2 g/t of Inferred material.

An oblique view of the HMB West design can be seen in Figure 15.19.

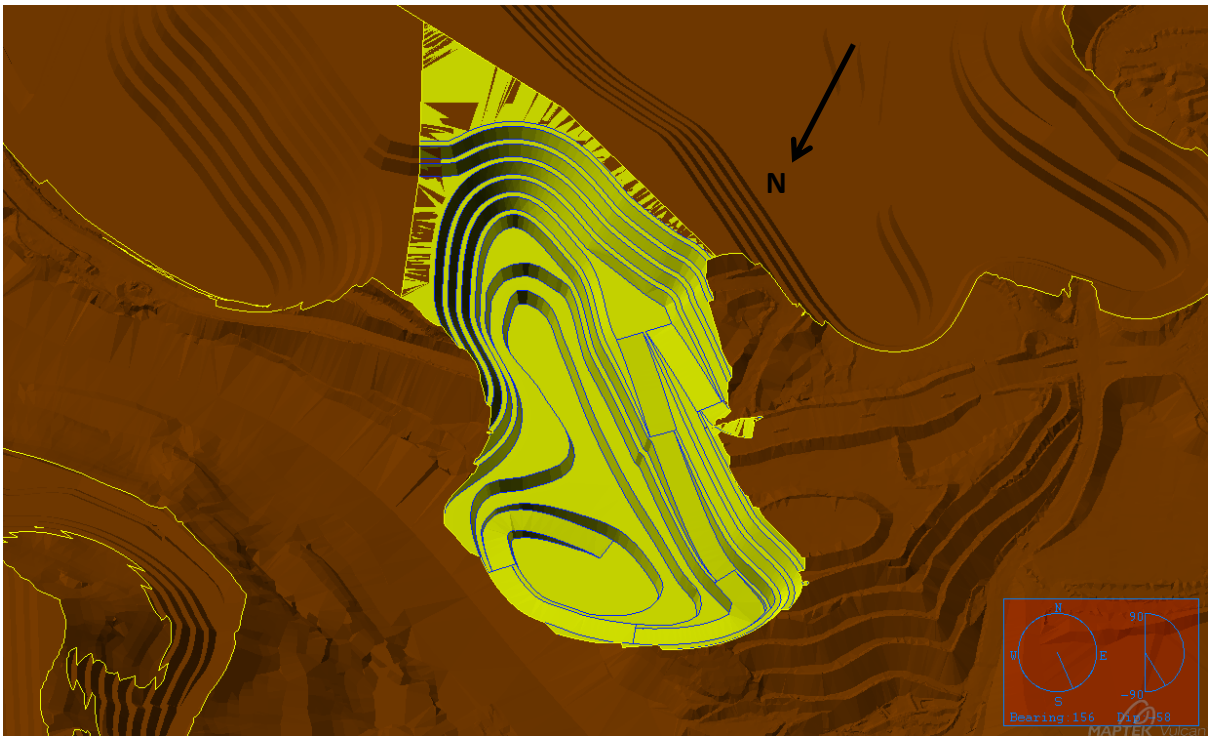


Figure 15.19 *Holy Moses - Basalt West pit design (view = bearing 156°, dip -58°), 2011 MGP mineral reserve estimate. (Golder 2012)*

15.4.6 Syndicate Design

The pit design parameters for Syndicate included:

- a single lane width of 15 m at a gradient of 1 in 10;
- in weathered material 10 m high batters at 55 degrees and 5 m berms,; and
- in fresh material 15 m high batters at 80 degrees and 8 m berms.

The Syndicate design only contains 0.6 Mt at 1.00 g/t Au of probable reserve (Table 15.21).

Category	Material type	Tonnage (Mt)	Gold grade (g/t)	Contained Au (000's oz)
Proven	Oxide	0.0	0.00	0
	Transition	0.0	0.00	0
	Fresh	0.0	0.00	0
	Total	**0.0**	**0.00**	**0**
Probable	Oxide	0.2	0.87	4
	Transition	0.1	1.07	5
	Fresh	0.3	1.03	10
	Total	**0.6**	**1.00**	**20**

Table 15.21 *Syndicate Reserve summary, 2011 MGP mineral reserve estimate.*

The estimated waste within the Syndicate pit is 2 Mt at a strip ratio of 2.6:1 t:t.

Contained within the Syndicate design is about 0.1 Mt at 0.8 g/t of Inferred material.

The Syndicate design angled image can be seen in Figure 15.20.

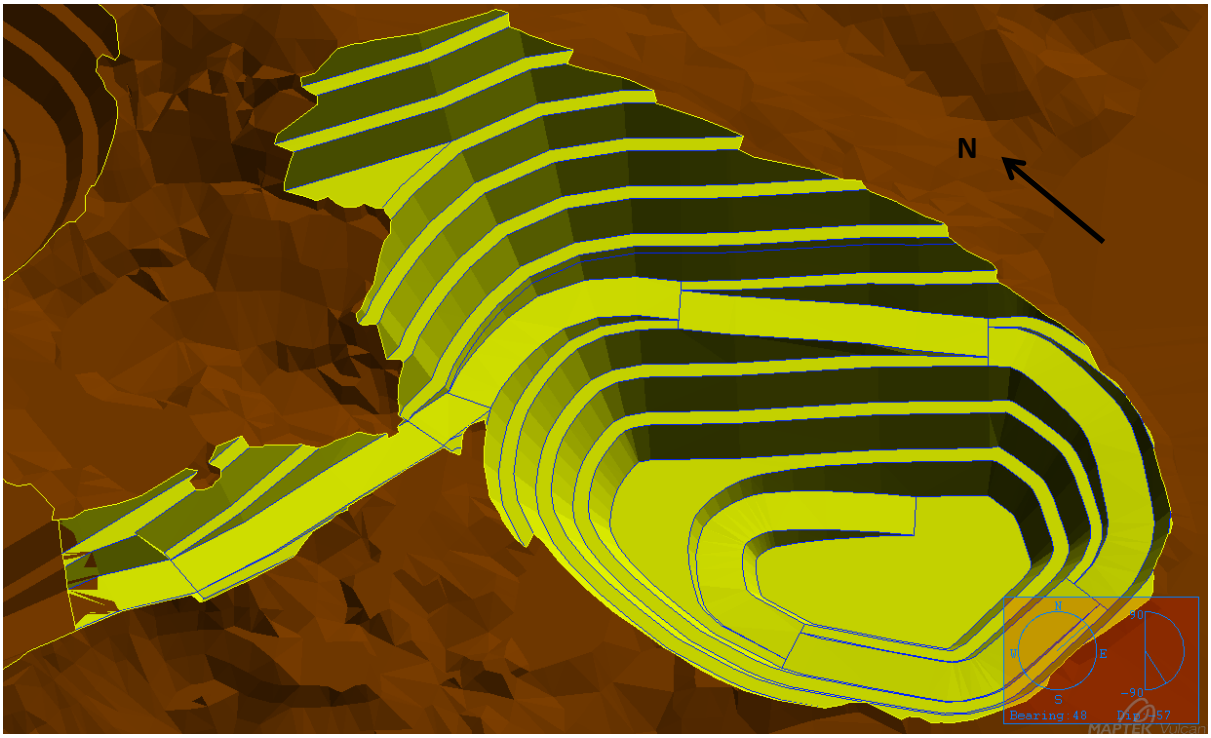


Figure 15.20 *Syndicate pit design (view = bearing 48°, dip -57°), 2011 MGP mineral reserve estimate. (Golder 2012)*

16 MINING METHODS

16.1 General

The MGP utilises conventional open pit mining methods. Mining is conducted by Leighton Contractors (Philippines) Incorporated (Leighton) under an alliance contract with FRC and PGPRC. Leighton is independent of CGA, FRC and PGPRC. In summary, mining consists of:

- RC grade control drilling – holes drilled at 60° incline perpendicular to the dominant strike of mineralisation over an interval of 10 vertical metres (Figure 16.1). The grade control pattern is standardised to a spacing of 12 m parallel to and 6 m perpendicular to the dominant strike of mineralisation. The grade control is pattern based on an optimisation study by Hellman & Schofield (2009). Samples are collected from every 1 m down hole interval from rig mounted rotary cone splitters. Samples analysed for gold content on site by SGS Limited using fire assay / AAS method.

Figure 16.1 *Grade control drilling, Colorado Pit, MGP 2012. Photograph taken looking northeast. (CGA 2012)*

- Blast hole drilling – RAB drilling of vertical drill holes over an interval of 5 vertical metres (+ subdrill of 0.5 m), using 115 mm diameter bits. Blast pattern spacing varies but typically has a burden of 3 m and spacing of 3.5 m. There is no sampling of blast holes – all grade control information is derived from RC drilling.
- Blasting – explosive supply and down hole loading is provided by Orica (Philippines) Limited, under sub contract to Leightons. Blasting typically uses emulsion product manufactured in a purpose built batching plant located on site. In some situations (pre-split, areas of excessive ground water) packaged explosive is substituted. Blasts typically use timings of 17 ms. Individual blast volumes vary but average 20,000 m^3. Blast heave typically averages 0.2 to 0.4 m in oxide / transition material and 1.0 m in fresh material.
- Loading – Ore and waste excavation is conducted by back hoe excavators over bench heights equating to in situ 2.5 vertical metres (Figure 16.2).

Figure 16.2 *Load and Haul operations, current mining fleet, Binstar deposit within Main Vein Pit, MGP 2012. Photograph taken looking east. (CGA 2012)*

- Hauling – conventional haulage utilising off highway rear wheel drive dump trucks.
- Mill feed –ore stockpile reclaim and plant feed methods utilising conventional front end loaders or direct dump from haul trucks.

Historical underground workings are present in some open pit mining areas (Binstar, Panique, Main Vein, Colorado). The location of underground workings is delineated from historical maps and plans in combination with targeted probe drilling and cavity surveying. Void delineation and working in the presence of known void areas controlled by documented policy including restricted access procedures. The void boundaries have proved conservative and regularly the void is not as large as has been modelled and / or backfilled with material of economic grade (Figure 16.3).

Figure 16.3 *Voids (white arrow) and back filled drifts (dashed lines) associated with historical underground workings, Binstar deposit within Main Vein Pit, MGP 2012. Photograph taken looking to the south. (CGA 2012)*

16.2 Pit Design Parameters

Pit wall designs have been developed in conjunction with independent geotechnical consultants (George, Orr and Associates) who regularly complete site visits to assess performance.

Routine geotechnical monitoring of interim and final pit walls is conducted. To date, mining has concentrated largely on the reduction of topographic highs and or stages within major pit designs and few final walls have been established. The performance of interim pit walls allows the assessment of various pit wall designs. Final walls have been established in small satellite pits (Montana, Libra East, HMB West) with good pit wall performance.

Limited hydrological investigation has been completed as mining has largely been conducted above the water table. Where required, active pit de-watering is conducted utilising manually controlled sump pumps. Dewatering has been required in HMB East staged pits and other satellite pits now completed (Montana, Libra East, HMB West). Surface water inflow is controlled by drainage developed in conjunction with the site water management plan.

16.3 Dilution and ore Loss

Routine measures taken to control mining dilution and ore loss during mining include supervision of ore zone mining activities by grade control personnel, floor level monitoring, blast movement correction of ore blocks and operator education.

16.4 Mining Equipment

Leighton operates and maintains the mining fleet which has been progressively changed out and upgraded with new equipment over the period March 2011 to May 2011. The current mining fleet consists of:

- 2 x Komatsu PC2000 back hoe excavators
- 1 x Komatsu PC1250 back hoe excavator
- 14 x Caterpillar 777 Dump Trucks
- 8 x caterpillar 773 Dump trucks
- 2 x Caterpillar D9 track dozers
- 2 x Komatsu D275 track dozers
- 4 x Caterpillar 988 Wheel Loaders
- 3 x Caterpillar 14M Motor graders
- 4 x Atlas Copco ECM660 top hammer blast hole rigs

16.5 Mine Production

Total movement (ore + waste) for the period April 2009 to March 2012 is illustrated in Figure 16.4. Total movement progressively ramped up to 700,000 BCM per month via the addition of additional mining fleet over the period April 2009 to December 2009 as mill throughput increased from 4 Mtpa to 6.5 Mtpa (Figure 17.3). Progressive change out and upgrade of mining fleet took place over the period March 2011 to May 2011 to meet long term planning requirements at 6.5 Mtpa processing rates.

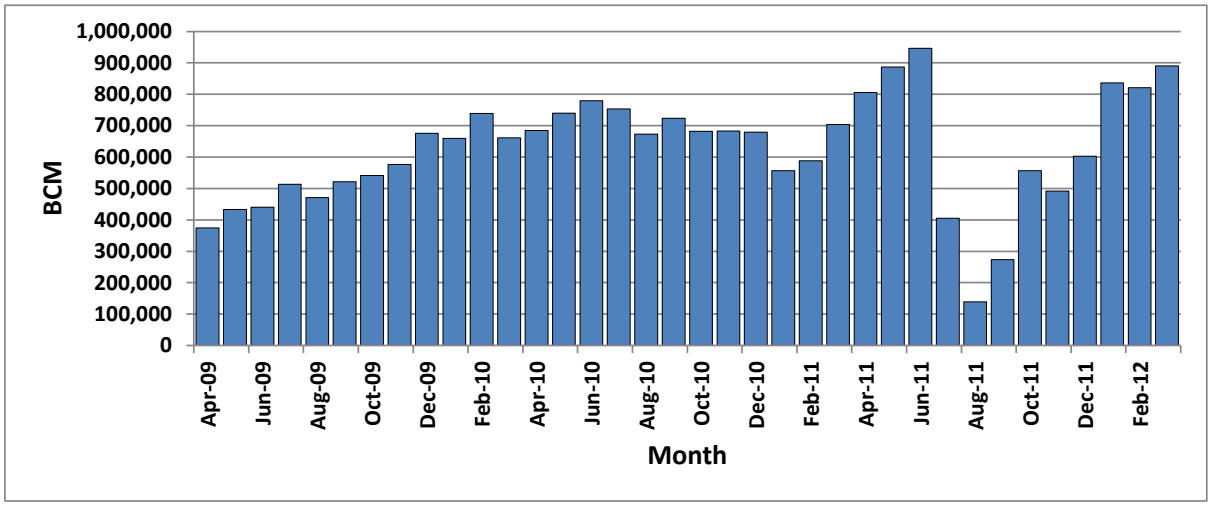


Figure 16.4 *Monthly total mining movement (BCM) over the period April 2009 to March 2012. Current mining fleet progressively commissioned over the period March 2011 to May 2011. Mining rates for July 2011 to December 2011 reflect failure of SAG Mill in the Masbate Processing plant and a corresponding reduction in mining movement to match reduced mill throughput (refer to Section 17.3). (CGA 2012)*

Life of mine (LOM) planning is based around total movement rates of 10 MBCM per annum, corresponding to 850,000 BCM to 900,000 BCM per month. The current mining fleet is capable of achieving LOM total movement. Measures to improve productivity, including increasing blast interval from 5 to 10 m, are being considered.

There are no waste pre-strip requirements for any current or future pits. LOM waste scheduling includes back fill of completed satellite pits.

17 RECOVERY METHODS

17.1 General

Ore mined from the MGP is processed on site within a conventional CIL processing plant owned and operated by PGPRC (Figures 17.1, 17.2). The processing plant was designed by Lycopodium Limited ("Lycopodium") with a nameplate throughput of 4 Mtpa, and was constructed over the period September 2007 to April 2009 by Leighton Contractors Asia Limited ("Leightons"). Ore treatment commenced 16th April 2009, with the first gold poured on 12th May 2009. Commercial production declared 1st July 2009.

Successful ramp up of milling throughput through 2009 and early 2010 resulted in a revised throughput target of 6.5 Mtpa. On softer mill feed blends (those dominated by oxide material) 6.5 Mtpa throughput was achieved with minimal changes to process flow, but additional crushing capacity was identified as an issue for harder feed blends. A supplementary crushing circuit was commissioned in September 2011 to support mill throughput at 6.5 Mtpa on mill feed blends containing 65% primary material. General overviews of the plant are illustrated in Figures 17.1 and 17.2.

Figure 17.1 *Photograph of Masbate Process Plant taken adjacent to feed level of the Supplementary crusher (right), looking west. SAG Mill visible in mid centre, with CIL tanks behind. Chimneys reaching to the horizon in centre right are associated with the 30 MW power station. (CGA 2012)*

Figure 17.2 *Photograph of Masbate Process Plant, looking south. Reagent storage and handling facilities visible in left foreground. CIL tanks mid centre. 30 MW power station (with chimneys) visible mid right. (CGA 2012)*

17.2 Plant Description

A Schematic diagram of the current plant flow sheet at 6.5 Mtpa throughput is presented in Figure 17.3.

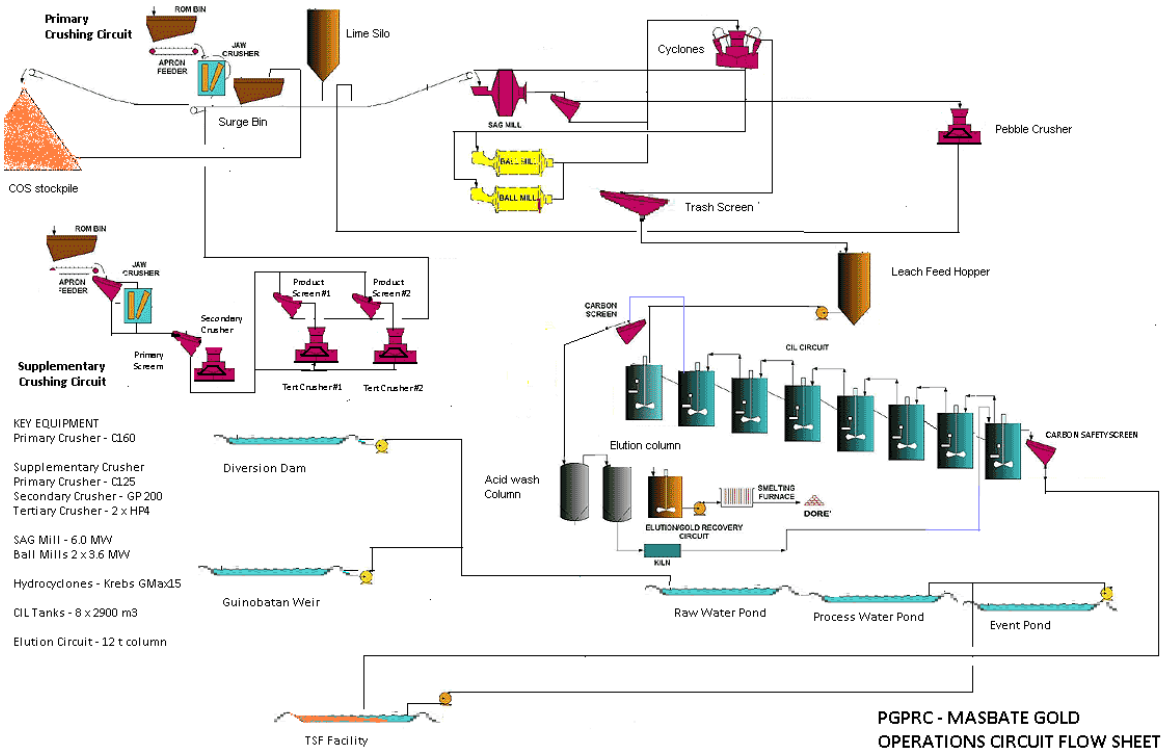


Figure 17.3 *Process flow sheet, Masbate Process plant, MGP. (CGA 2012)*

In detail the plant comprises:

- Primary single stage jaw crusher (Nordberg C160) which reduces ore to a top size of 150 mm. The product from the primary crusher is fed directly to the SAG Mill. A diverter gate on the conveyor system allows construction of a crushed ore stockpile and provides continuity of feed to the grinding circuit when the jaw crusher requires routine maintenance.
- Supplementary 3 stage crushing circuit consisting of a single Nordberg C125 jaw crusher feeding a secondary Nordberg GP200 cone crusher and two tertiary Nordberg HP4 cone crushers. The supplementary crushing circuit reduces a portion of run of mine ore to a 12 mm product which flows through the SAG Mill to the Ball Mill section of the grinding circuit.
- A single 6.0 MW Koppers SAG Mill receives feed from the primary crusher. Underflow discharge of product less than 12 mm diameter is directed to the Ball Mills. Overflow product is recycled back to SAG feed after passing through a pebble crusher circuit.
- Two parallel 3.6 MW Ball Mills to produce a product of P_{80} at 106 micron.
- Leaching of gold using cyanide in a conventional 8 x 3000 m^3 tank circuit.
- Recovery of gold from loaded carbon using a normal elution and single pass electrowinning circuit, followed by smelting to produce gold bullion.

17.3 Plant Production History

To the end of March 2012, the Masbate plant has processed 13.6 Mt of material at an average grade of 1.16 g/t to produce in excess of 432,000 oz Au since the commencement of commercial operation in July 2009.

Process availability averages 89.7% for the period July 2009 to March 2012 (Figure 17.4). The saw tooth pattern in availability generally reflects quarterly replacement of SAG Mill liners. Systematic trialling of SAG Mill liner design has been ongoing since plant commissioning and resulted in significant improvements in time to replacement.

Process recovery averages 84.9% for the period July 2009 to March 2012 (Figure 17.5). Overall process recovery reflects both the composition of the mill feed in terms of material type (oxide, transitional, primary) and residence time within the CIL circuit (refer to Section 13). Design residence time at 4 Mtpa is 24 hours. At 6.5 Mtpa throughput, residence time within the CIL circuit is reduced to 19 – 20 hours. Addition of Lead Nitrate has enabled the process to maintain design recovery, and for some ore types (transitional and primary) improve on design recovery, at 6.5 Mtpa throughput rates by driving leach kinetics hard in the first 3 to 6 hours.

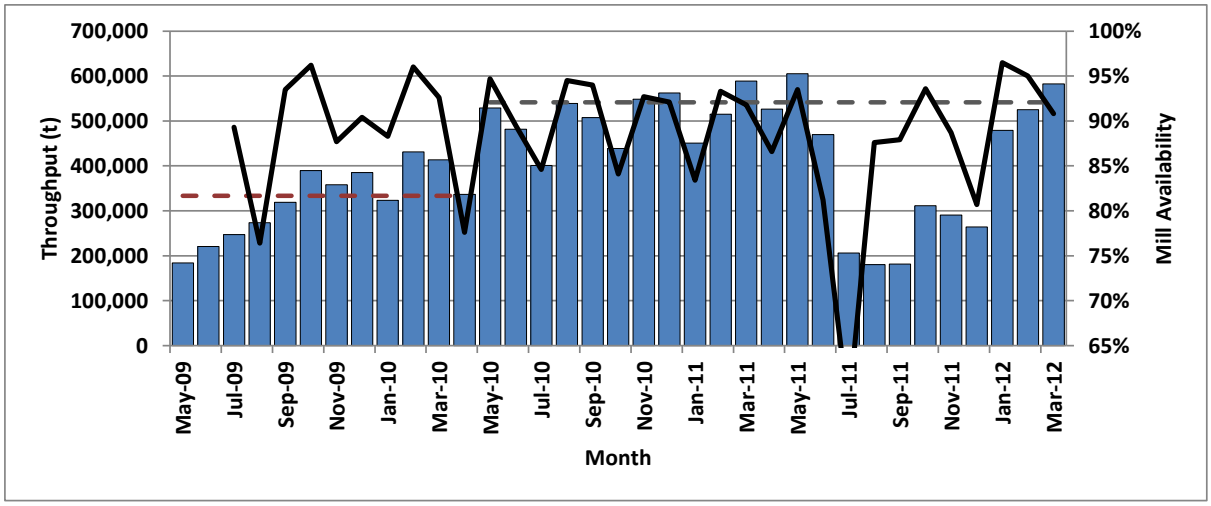


Figure 17.4 *Monthly throughput (bars; left hand axis) and availability (solid black line; right hand axis) for the Masbate Processing Plant. Dashed red line equates to annualised throughput of 4 Mtpa; Grey dashed lines equates to annualised throughput of 6.5 Mtpa. Reduced throughput for the period July 2011 to December 2011 was due to SAG Mill repairs. (CGA 2012)*

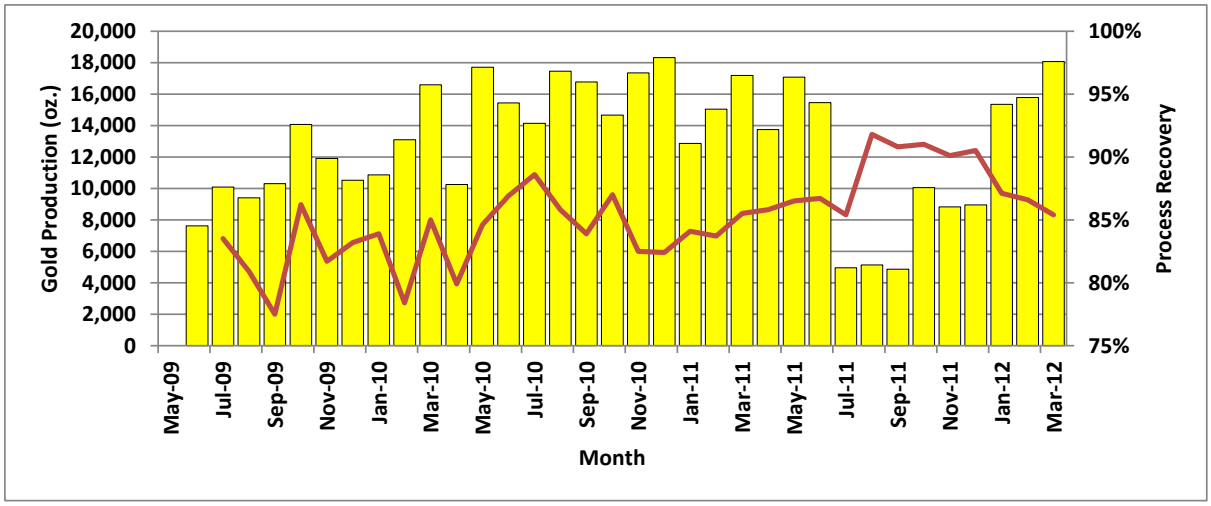


Figure 17.5 *Monthly gold production (bars; left hand axis) and process recovery (solid brown line; right hand axis) for the Masbate Processing Plant. Reduced production for the period July 2011 to December 2011 was due to SAG Mill repairs. Sharp dips in recovery during February 2010 and April 2010 reflect lack of lead nitrate reagent due to interruption to delivery to site. (CGA 2012)*

17.4 Plant Production Planning

Forward planning of process plant production targets a throughput rate of 6.5 Mtpa.

Systematic improvements in mill linear design and materials have resulted in improved mill availability as operational experience builds. Forward planning assumes a 6 month cycle of: 5 months @ 94% Mill Availability followed by 1 month @ 87% Mill Availability (average of 92.8%).

Major reagent and power consumption rates are related to physical material type of ore processed. Table 17.2 details typical consumption rates for major consumables and includes values used in forward planning.

Consumable Item (per tonne processed)	Monthly rate	Forward Planning		
		Oxide	Transition	Primary
Power (kWh)	14.5 – 16.5	12.1	14.8	17.3
Cyanide (kg)	0.53 – 0.68	0.53	0.58	0.68
Lime (kg)	2.20 – 2.70	2.70	2.50	2.20
SAG Grinding Media (kg)	0.13 – 0.15	0.13	0.15	0.15
Ball Mill Grinding Media (kg)	0.42 – 0.45	0.42	0.42	0.45
Lead Nitrate (kg)	0.13	0.13	0.13	0.13

Table 17.1 *Major consumable consumption rates and values used in forward planning, Masbate Process Plant.*

18 PROJECT INFRASTRUCTURE

18.1 General

The Masbate Gold Project has a well-developed infrastructure associated with mining and processing operations. Infrastructure is a mixture of refurbished items originally constructed during previous mining operations by Atlas (1980 – 1994) and new facilities constructed since 2009. The location of significant mining infrastructure is illustrated in Figures 18.1 and 18.2.

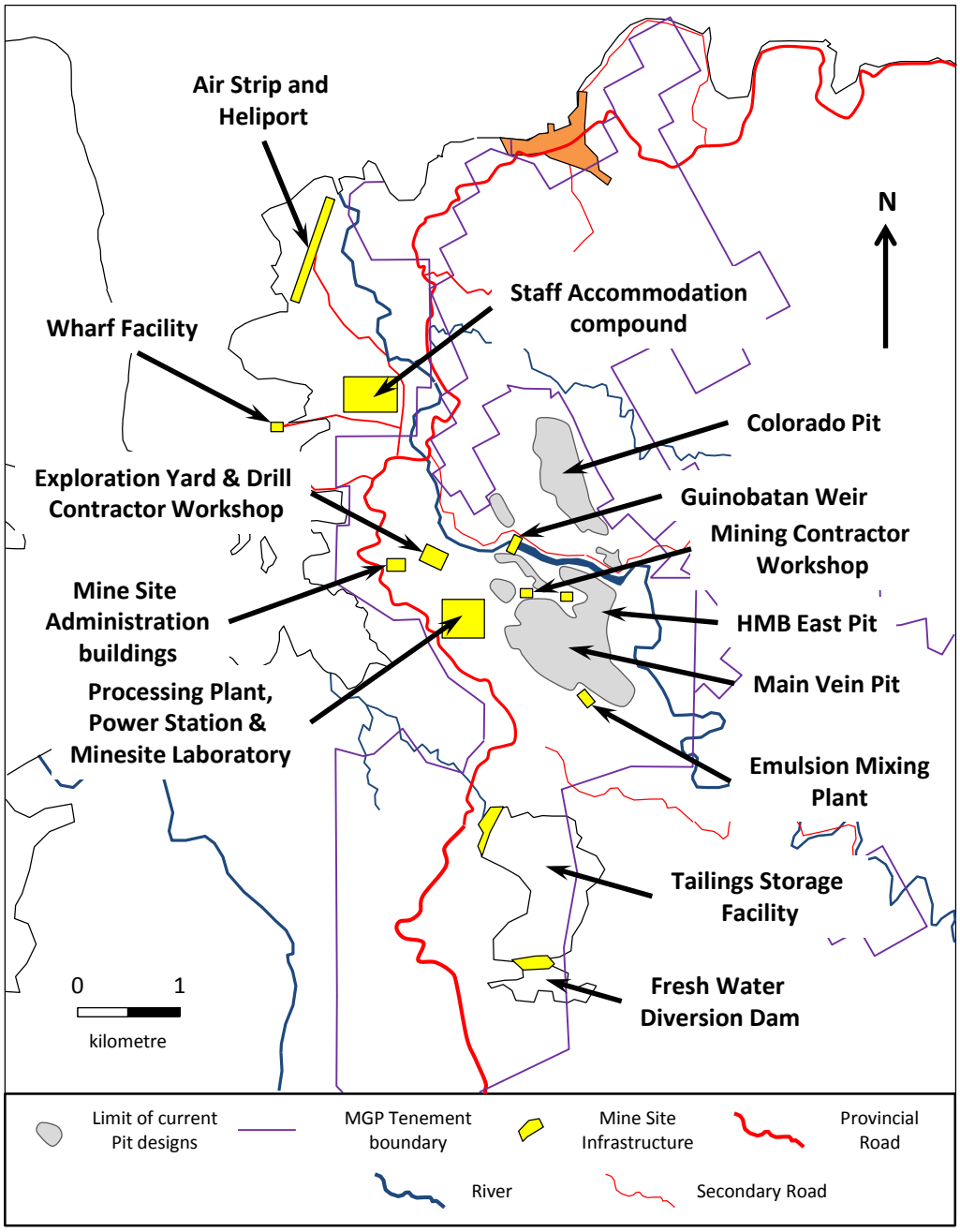


Figure 18.1 *MGP Infrastructure. (CGA 2012)*

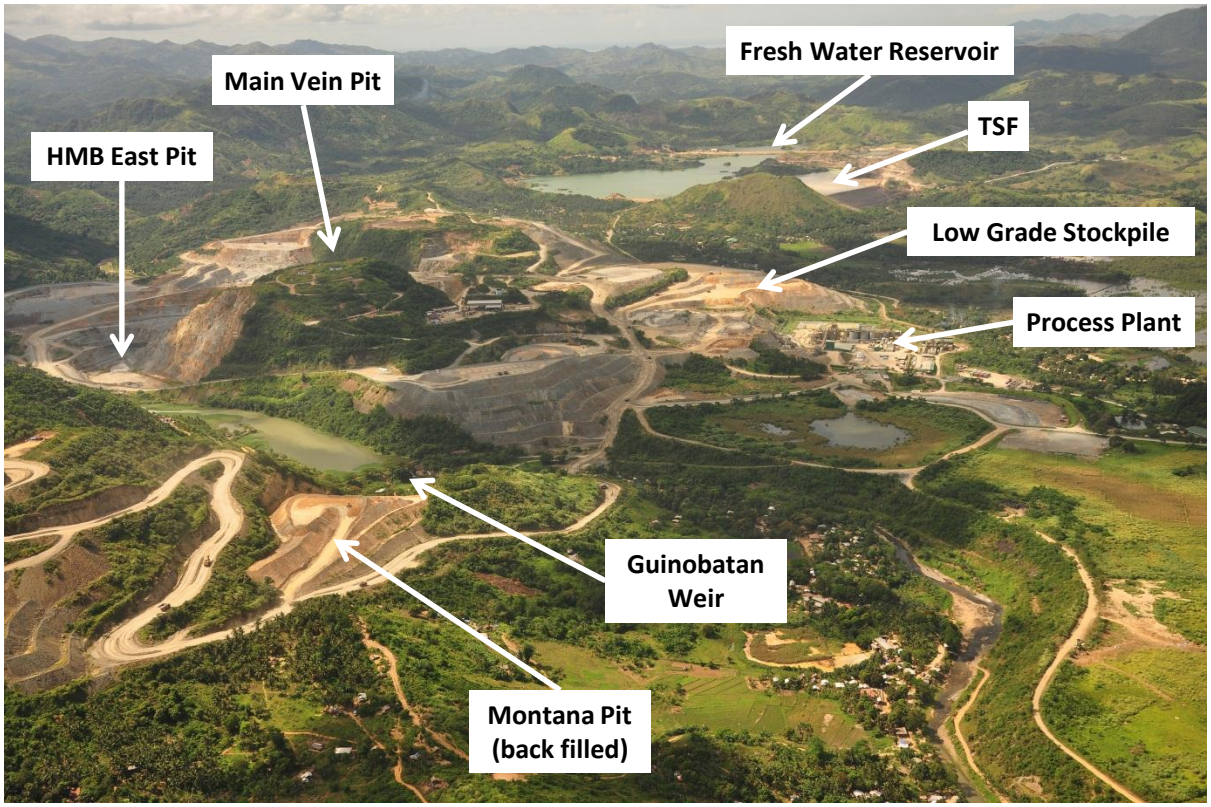


Figure 18.2 *MGP Infrastructure. Photograph looking to the south. (CGA 2012)*

18.2 Roads

A series of well-maintained metal roads access the Masbate Process Plant and Mine Site Administration buildings from the property boundary and link into the Provincial road network at controlled points.

Open pits, dumps and stockpiles are accessed by a network of haul roads constructed and maintained to accommodate passage of Cat 777 Dump trucks (Figure 18.3).

Exploration areas within the property are serviced by a series of secondary haul roads and tracks constructed to allow passage of track mounted drill rigs and light vehicles. In some instances exploration areas are accessed from the provincial road network.

Figure 18.3 *Colorado Haul Road including Guinobatan River crossing, MGP. Guinobatan Weir located upstream of the haul road crossing (centre right). Buildings in lower left are Goldbag, part of the Syndicate Barangay. Photograph taken looking north east. (CGA 2012)*

18.3 Port Facilities

Private wharf facilities located at the Masbate Gold Project can accommodate loading / unloading of two standard 500t flat bottom barges at high tide (Figure 18.4). The port also provides permanent mooring for the Lady Jacqueline, a converted fishing trawler owned by Leightons and utilised for passenger transport to / from commercial ports at Masbate City or Pilar. The port is permitted by the Philippines Port Authority (PPA).

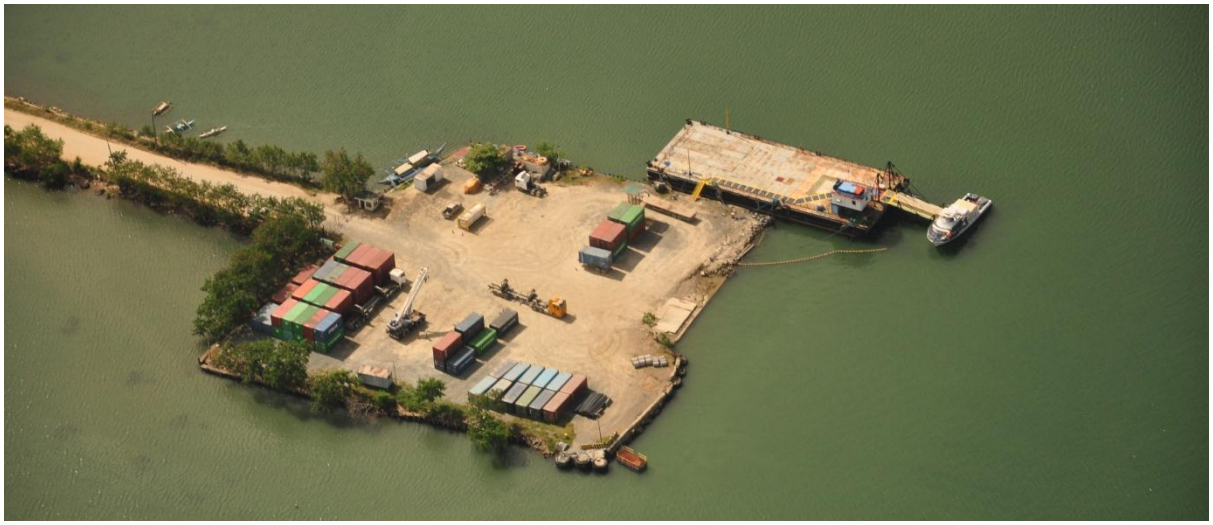


Figure 18.4 *Causeway port facilities, MGP. (CGA 2012)*

18.4 Air Strip

The Masbate Gold Project operates a private 900 m unpaved airstrip (Figure 5.1). Private charter flights between Masbate Gold Project and Manila utilise fixed wing aircraft that can accommodate up to 19 passengers. The air strip is permitted by the Civil Aviation Authority of the Philippines (CAAP).

18.5 Waste Dumps and Ore Stockpiles

Waste dumps are located close to active mining areas. Several low grade stockpiles are located near to the Processing Plant (Figure 18.2).

18.6 Tailings Disposal

Tailings generated from processing Masbate ore are deposited in a Tailings Storage Facility (TSF) located 2.5 km south of the Masbate Process plant (Figure 18.1, 18.2, 18.5). The TSF is designed by GHD, a leading international engineering, architecture and environmental consultancy company, and incorporates provision for local climatic (typhoon) and seismic conditions. The design consists of a main dam wall and subsidiary walls to be constructed in series of lifts.

Initial TSF construction began in 2008 and is ongoing. Construction is conducted by earth moving contractors (Galeo Equipment and Mining Company Inc.) under supervisor and direction by FRC. GHD regularly monitor construction against design and submit reports to both FRC and the MGB. Independent 3rd party inspections are also completed at the request of the MGB (AMH Philippines Inc.) and FRC (DE Cooper & Associates Pty Ltd).

The TSF is permitted to a final height of 51 m RL, including a final 2 m freeboard provision (i.e. final tails deposition height of 49 mRL). At completion the TSF has a storage capacity of 61.7 Mt. At the end of March 2012, the dam wall was at a height of 38 m RL.

A section of the Masbate provincial road network lies within the footprint of the TSF (Central Nautical Highway). A replacement for this section of road has been constructed, approved and handed over to the Philippines government – this process was completed in early 2012.

Figure 18.5 *Tailings storage facility (TSF), MG. Photograph taken looking north west. (CGA 2012)*

Tailings deposition within the TSF is controlled in such a way as to minimize water contact against constructed dam walls. Water is recycled back to the Masbate Process plant from the TSF and currently accounts for at least 75 % of process water needs.

18.7 Water

FRC holds permits to draw water for industrial use from the Guinobatan, Bangyon and Lanang Rivers. To date, total process water requirements have been drawn from the Guinobatan River only, at a weir constructed during the Atlas period of operations (1980 to 1994). Domestic water for the Masbate Gold Project is drawn from deep wells located close to the Guinobatan Weir. Water permits are administered by the Nation Water Resources Board (NWRB).

18.8 Power

Electrical power for the Masbate Gold Project is supplied from a purpose built 30 MW power station located immediately adjacent to the Processing Plant. The power station consists of several generating sets operating on heavy fuel oil (HFO). MGP is currently licenced to operate a power station up to 50 MW generating capacity.

19 MARKET STUDIES AND CONTRACTS

19.1 MGP Management

The MGP is owned and operated by FRC, PGPRC and VMC.

Control of various areas and facilities of the MGP as well as the roles and responsibilities of FRC and PGPRC in the ongoing management of the project is contractually defined in an agreement between the two parties dated 5th July 2001.

PGPRC is contractually required to process all ore mined by FRC under a separate ore sales agreement executed on the 5th July 2007 and applicable for the life of the mining operation. Under the terms of the agreement, PGPRC purchases ore from FRC at the point of loading for total mining cost plus a margin, the magnitude of which is proportional to the Au content of high grade ore received during each month.

Under additional contracts which have been negotiated, agreed and in the final stages of legal drafting, VMC delegate management of MPSA-219-2005-V to FRC via a technical services agreement, and VMC agrees to supply ore mined from MPSA-219-2005-V to PGPRC via an ore sales agreement similar in structure the existing ore sales agreement between FRC and PGPRC.

19.2 Mining

Leighton Contractors (Philippines) Incorporated provide a comprehensive construction and open pit mining service to the Masbate Gold Project under an industry standard alliance contract with FRC and PGPRC which was executed 20th March 2008. Stage 2 (mining operations) of the contract commenced on 1st September 2008 with a period of 6 years. With progressive upgrade of the mining fleet, the contract was recast from 1st March 2011 for a period of 6 years.

The alliance contract sets prices for a variety of fixed and variable items plus a margin, the magnitude of which is related to monthly performance against a number of agreed KPI measures. The contract operates under terms and conditions which generally comply with industry norms and allows flexibility to replace or further increase mining fleet as required.

19.3 Power Supply

Power station operation and maintenance is conducted by Wartsila Philippines Incorporated under an agreement with PGPRC. The agreement sets pricing for both a fixed monthly fee and a variable component related to generating set engine hours. The agreement operates under terms and conditions which comply with industry norms and was originally executed on the 27th February 2009 for a period of 3 years. The agreement is currently under renegotiation for a further 3 year period.

19.4 Refining

A contract is in place with Metalor Technologies SA for the transportation and refining of dore bullion into fine gold and silver for sale. This contract sets prices for transporting and refining the dore under terms and conditions which generally comply with industry norms. The contract was initially executed on the 8th July 2009 and has subsequently rolled over annually on the same terms and conditions.

19.5 Project Financing

CGA secured an USD 80.3M project finance facility by BNP Paribas, which was fully drawn during the 2009 financial year to fund development of the MGP. As of March 31st 2012, USD 48.9M had been repaid and the loan facility had an outstanding balance of USD 31.4M.

19.6 Gold Hedging and Forward Sales

A hedging program of puts covering 46,079 ounces of Gold and forward sales covering 214,337 ounces of gold was executed during the September 2008 quarter. At 1st April 2012 the remaining balance of forward sales contracts was 93,914 ounces at gold at an average price of US$899, with no puts outstanding.

Forward gold sales commitments are approximately 4,800 oz. per month and scheduled to be completed in December 2013.

19.7 Authors Opinion

The qualified person has reviewed the agreements outlined in this section and confirms they support the assumptions of this Technical Report.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 Regulatory Framework

MPSA's are granted or entered into on the condition that the subject mining activities are managed in a technically, financially, socially, culturally and environmentally responsible manner.

The DENR requires an Environmental Compliance Certificate (ECC) for any mining activity except when an EP has been issued or during the exploration stage of an MPSA. The ECC is issued by the DENR based on an Environmental Impact Statement (EIS) prepared by the company in accordance with procedures under the ECC system. An ECC obliges a company to submit and implement an Environmental Protection and Enhancement Programme (EPEP) for the life of the mine. The EPEP forms the parent document for the development and implementation of Annual Environmental Protection and Enhancement Programmes (AEPEP) which require approval from the DENR.

The ECC system regulates a funding structure to ensure company compliance with EPEP commitments and ensure immediate funding is available for rehabilitation in the event of environmental damage during mining operations. The structure of the Contingent Liabilities and Rehabilitation Fund (CLRF) is illustrated in Figure 20.1.

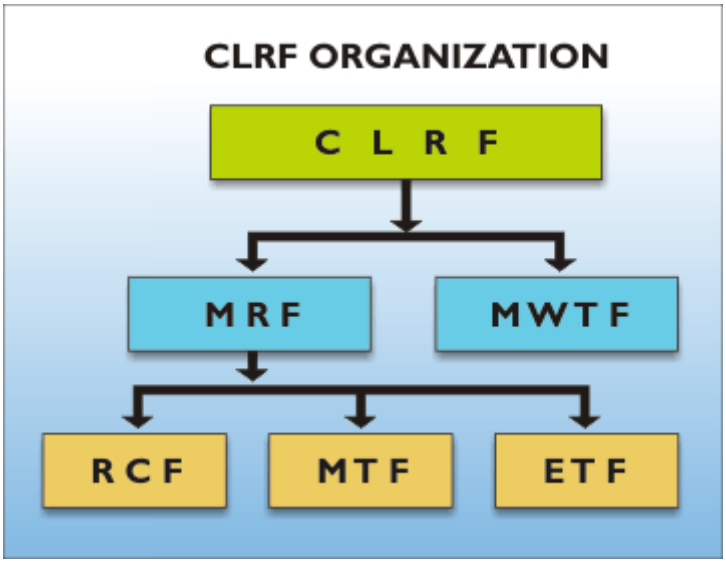


Figure 20.1 *Contingent Liabilities & Rehabilitation Fund (CLRF) structure. MRF = Mine Rehabilitation Fund; MWTF = Mine Waste & Tailings Fees; RCF = Rehabilitation Cash Fund; MTF = Monitoring Trust Fund; ETF = Environmental Trust Fund. (DENR 2012)*

Mine Waste & Tailings Fees (MWTF) are set are PHP 0.05 per tonne of mine waste and PHP 0.10 per tonne of tailings generated by mining activity.

The Mine Rehabilitation Fund (MRF) has three components:

- Rehabilitation cash Fund (RCF) consisting of a one off payment of PHP 5M
- Monitoring Trust Fund (MTF) equating to ongoing payments of PHP 50,000 per month (refer to section 20.5)
- Environmental Trust Fund (EFT) consisting of PHP 50,000 initial payment, topped up as required.

CLRF funds are held in a Philippine government deposit account and administered by the DENR.

20.2 MGP Environmental Permits

An EIS was prepared for the MGP by Base Metals and submitted to the DENR in March 1998. The DENR issued an ECC (9804-003-12C) for the project in June 1998. The ECC pertains to the mining properties within an area of 443 ha and a project involving mining, process plant, power plant, new TSF and other support facilities. The ECC permits open pit mining with a production capacity of twenty thousand (20,000) tonnes per day utilising CIL the benficiation process.

In 2005 FRC updated baseline studies in the 1998 EIS and renewed the ECC for the MGP. A comprehensive EPEP was prepared, submitted and accepted by the DENR.

As of 31st March 2012, the balance of CLRF associated with the MGP is as follows:

- RCF = PHP 5.865M
- MTF = PHP 0.180M
- EFT = PHP 0.052M

20.3 Environmental Studies

Comprehensive environmental studies within and adjacent to the mining tenements of the MGP were conducted in 1997 – 1998 (EIS preparation) and 2004 – 2005 (EPEP preparation), prior to the commencement of mining operations. The studies noted that the natural environment of the project area has been highly modified, and in some instances polluted, by human land use which is dominated by agriculture and small scale mining activity.

Monitoring established baseline environmental conditions in terms of water quality (surface water and groundwater), sediment quality, air quality and noise.

The surface waters within and adjacent to the project area (Guinobatan River, Panique Estuary, Port Barrera) are slightly alkaline (pH 7 to 9). Baseline sediment monitoring indicated that small scale mining has had a detrimental impact on sediment quality of the Guinobatan River. Mercury was present above detection limits in sediment samples collected at Guinbatan River, Panique and Port Barrera. Additional sediment samples from Port Barrera and the Panique Estuary had mercury levels above trigger levels of Australian and New Zealand Guidelines for Fresh and Marine Water Quality (ANZECC 2000 – no Filipino guidelines available at the time of analysis).

Noise monitoring levels at locations near the MGP prior to operations were typical of rural area. Within local settlements, moderate noise levels were common with noise generated in part from traffic activity, local small scale ore processing plants (rod and ball mills) and generators.

Ambient air quality monitoring established the baseline concentrations of air pollutants prior to the operation of the Masbate power station. Parameters monitored included NO_2, SO_2 and total suspended particles (TSP). Overall, the air quality of the MGP prior to operations was satisfactory and typical of that for a rural area.

Flora and fauna surveys were conducted in terrestrial, freshwater and marine environments. Very little of the vegetation within the studied area was of significant commercial or environmental

significance. Fauna studies documented 37 species of insects, 31 bird species, 7 mammal species and 1 species of reptile. It was determined that the MGP would not significantly affect the habitat of the 5 species of migratory birds that have been documented in the area. Some mangrove habitat has been severely affected by local fish and prawn farming activity and there was evidence of further mangrove degradation due to timber cutting and charcoaling.

Desk top research and field visits found no significant archaeological resources of sites at risk from the MGP and an archaeological clearance was issued by the Philippine authorities.

20.4 Waste and Tailings Disposal

Tailings disposal takes place in an engineered TSF (section 18.6).

Waste management policies implemented on site utilise the principles of reuse and recycle. The MGP manages a landfill facility that also accepts domestic refuse from local barangays. Hazardous waste including hydrocarbons (oil and lubricants), cyanide packaging and batteries are collected from site by operators with registered recycling facilities.

The MGP complies with the guidelines of the International cyanide convention.

20.5 Site Monitoring

Under the terms of the ECC, a multipartite monitoring team (MMT) composed of representatives from the MGB Region V Office, DENR Region V Office, local environmental NGO, affected communities and FRC shall be organised to serve as the monitoring arm of the CLRF committee. The MMT conducts a quarterly site visit to the MGP and measures the performance of the operation against conditions of the ECC and approved AEPEP. Financing of the activities of the MMT comes from the MFT (refer to section 20.1).

DENR officials conduct routine inspections and audits of the operation.

FRC and PGPRC conduct routine environmental monitoring including monthly surface water and sediment chemical analysis, noise monitoring and air quality measurement. Annual ambient noise and emission testing is also conducted at the power station. Results of site environmental monitoring are made available to the DENR. Periodic ecological surveys are conducted within the freshwater, marine and coastal environments. Funding of FRC and PGPRC environmental sampling is separate to the CLRF and is assigned from operating budgets.

20.6 Water Management

The MGP has developed a site water management plan. Under the terms of the ECC, the site may discharge water offsite provided the quality of such is equal to or better than the receiving body(ies) of water. Pit and surface run-off water quality is controlled through silt ponds and vegetation buffers. The TSF has freeboard requirements that can accommodate extreme storm inflows.

There are sufficient local water sources available to meet all site process and domestic water requirements. More than 75 % of process water is recycled from the TSF.

20.7 Community Development

The Philippines Mining Act (1995) mandates mining lease holders to effect development in the host and neighbouring communities based on the communities' own identified programmes and projects through a Social Development and Management Plan (SDMP). Under an amendment to the act, annual SDMP commitments must equate to 1.5 % of total operating costs, though the MGP has an existing agreement with DENR whereby SDMP commitment is capped at 1 % of direct mining and milling costs for a five year period following the commencement of commercial production (1st July 2009).

MGP works closely with elected officials and community leaders of eight local barangays (native Filipino word for the equivalent of village, district or ward), commonly referred to as the "impact barangays" (Figure 20.2, refer to section 5.4). The impact barangays are: Amoroy, Bangon, Puro, Syndicate, Panique, Capsay, Balawing and Syndicate, with a total population of approximately 24,000 people. Annual SDMP programmes are developed in conjunction with the leaders of each impact barangay and submitted to the DENR for approval. Assistance to date has largely concentrated on infrastructure upgrade projects, health and education initiatives, and programmes associated with livelihood and cultural activities.

In addition to SDMP, the MGP has implemented a Corporate Social Responsibility (CSR) programme whereby additional funding is made available to community initiatives. Projects under CSR generally have a broad community focus, may extend beyond the borders of the eight impact barangays and are not subject to prior approval by the DENR.

Total annual expenditure on SDMP and CSR programmes by the MGP is around USD 750,000. Funding, project development and implementation is administered by the community relations department of the MGP, who also continually inform and educate the community of current mining and exploration activities.

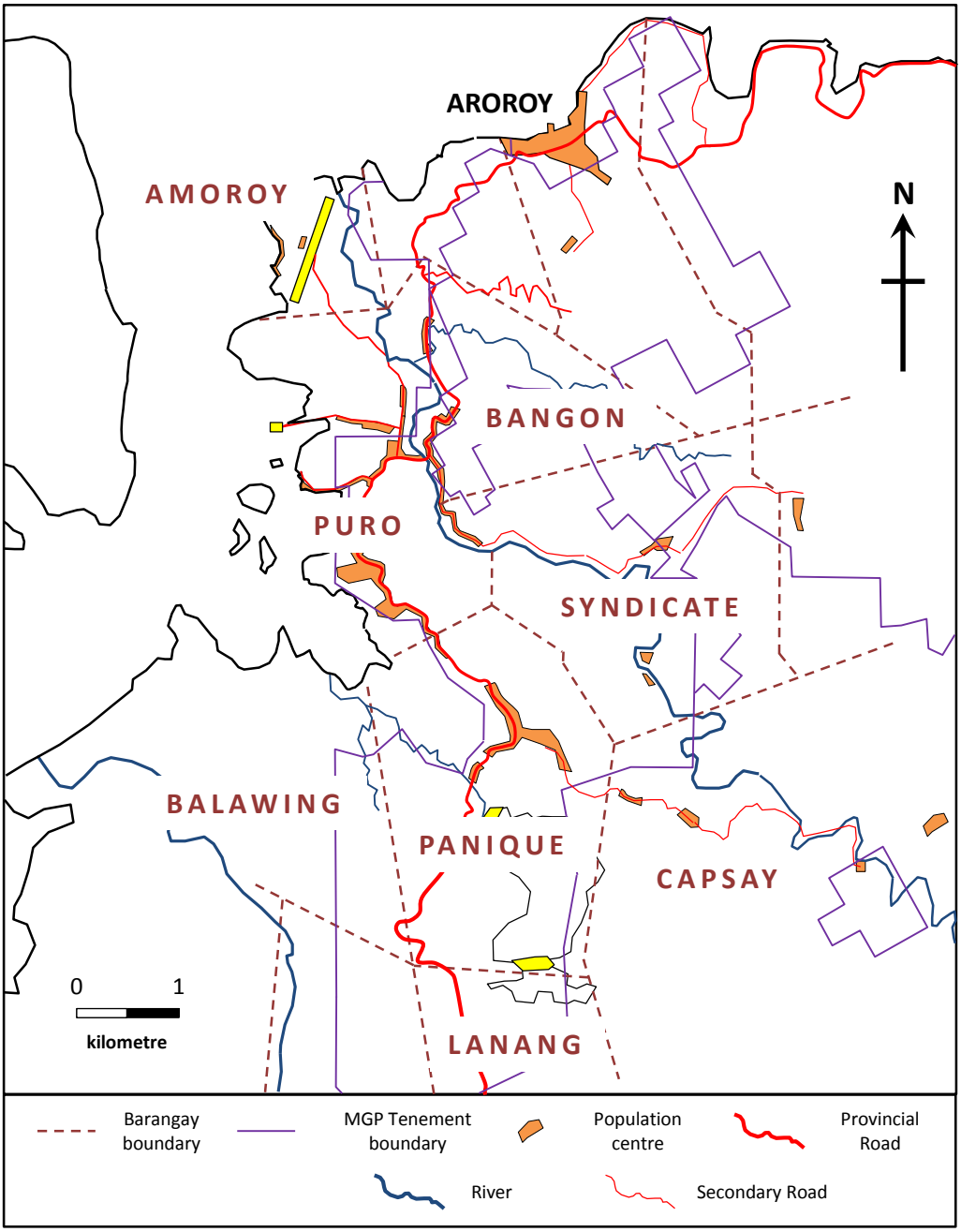


Figure 20.2 *Location of the eight impact barangays, MGP. (CGA 2012)*

20.8 Mine Closure Planning

Mine closure planning is included within the EPEP and approved by the DENR. A Final Mine Rehabilitation & Decommissioning Fund (FMRDF) has been established. The FMRDF has a total value of PHP 101.6 M and will build up over a five year period from 2013 to 2017 (Table 20.1). The FMRDF, which is separate from CLRF (section 20.1) is held in a Philippine government deposit account and administered by a committee including DENR officials.

Year	Deposit (PHP M)	Total (PHP M)
2012	0	0
2013	37.3	37.3
2014	30.5	67.8
2015	18.3	86.1
2016	12.5	98.6
2017	3.0	101.6

Table 20.1 *Final Mine Rehabilitation & Decommissioning Fund payment schedule, MGP.*

21 CAPITAL AND OPERATING COSTS

21.1 Operating Costs

Adjusted operating cash costs on a USD/oz and USD/tonne milled for the July 1 2009 to June 30 2010 and July 1 2010 to June 30 2011 financial years are presented in Table 21.1. Cost escalation was largely driven by price increases for major reagents (diesel, HFO, cyanide) over the period.

Adjusted Cash Cost	2010 FY	2011 FY	Projected
USD / oz	550	641	700
Mining (USD / t milled)	6.19	6.91	8.3
Processing (USD / t milled)	7.48	8.50	10.3
Administration (USD / t milled)	4.68	4.48	4.3
Total (USD / t milled)	18.35	19.89	22.9

Table 21.1 *Historical adjusted operating costs, MGP.*

Projected operating costs for LOM planning have been derived from current costs, using known contract rates, manning levels, power, reagent and consumable usage rates and maintenance spares costs. Future rates for salaries and prices for reagents, consumables and mechanical spares have been estimated.

Higher projected costs reflect a reduction in head grade in remaining ore reserves compared to processed material in the 2010 and 2011 financial years. Initial mining at the MGP concentrated on the early completion of higher grade satellite deposits (Montana, Libra, Holy Moses – Basalt).

21.2 Sustaining Capital Planning

CGA has developed a preliminary capital expenditure programme for MGP to 2017 based on continued operations at current production levels (6.5 Mtpa processing rate). The programme (Table 21.1) includes a range of expenditures which are essential to the continuity of the operation, required for regulatory compliance or required to maintain plant conditions at satisfactory levels.

Capital Project	Value (USD M)	Indicative timing
Mining Equipment lease (including interest)	8.0	Annual to 2017
Ongoing TSF expenditure	6.0	Annual to 2017
Sustaining Capital – Mining	2.0	Annual
Sustaining Capital – Processing	4.0	Annual

Table 21.2 *Sustaining capital schedule, MGP.*

Discretionary spending is not included in sustaining capital estimates. Discretionary spending items include continued exploration and resource definition (2012 Budget = USD 20M) or studies into options to improve economic return from the MGP.

22 ECONOMIC ANALYSIS

This Technical Report updates minerals resource and mineral reserve estimates for the MGP. It does not include provision for a material expansion of current production. An economic analysis is outside the scope of this report.

23 ADJACENT PROPERTIES

23.1 General

Mineral tenements surrounding the MGP are illustrated in Figure 23.1, with details provided in Table 23.1.

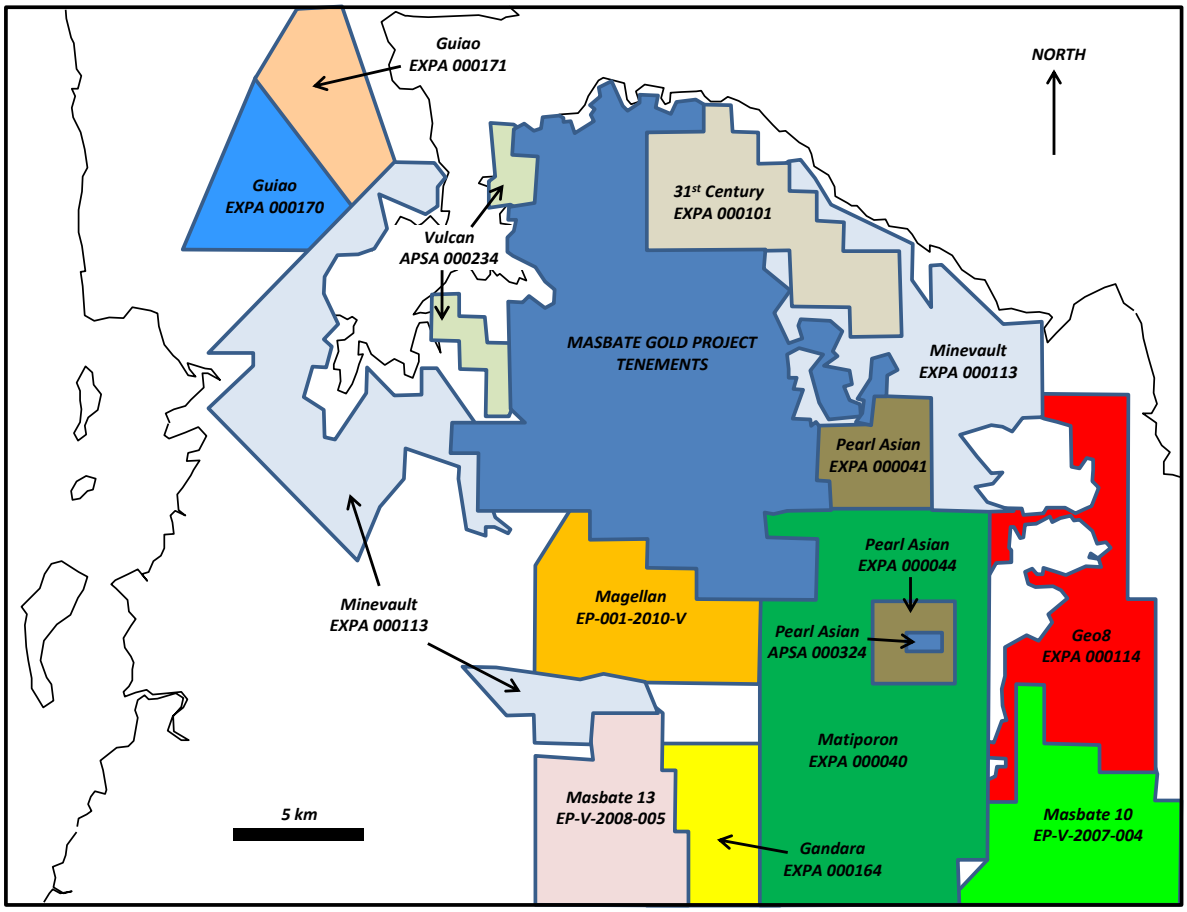


Figure 23.1 *Mineral tenements adjacent to MGP. Data sourced from DENR, May 2012. (CGA 2012)*

Tenement ID	Holder	Date	Area (ha)	Commodity
APSA-000234-V	Vulcan Industrial & Mining Corp.	09/16/97	764.00	Au, Cu
APSA-000324-V	Pearl Asian Mining Industries Inc.	09/29/04	72.00	Au, Cu, etc.
EP-V-2007-004	Masbate 10 Philippines Inc.	12/11/07	15,007.43	Au, Cu, Ag, etc.
EP-V-2008-005	Masbate 13 Philippines Inc.	2/7/2008	8,357.35	Cu, Au, etc.
EP-001-2010-V	Magellan Consolidated Mines & Development Corp.	7/30/2010	2,799.99	Au, Cu, Mn, etc.
EXPA-000101-V	31st Century Mining Corp.	09/14/07	2,754.67	Au, Cu, etc.
EXPA-000164-V	Gandara Minerals & Development Corp.	07/20/10	2,366.91	Au, Cu, etc.
EXPA-000170-V	Jennifer T. Guiao	10/08/10	1,619.76	Mn
EXPA-000171-V	Joseller Yeng Z. Guiao	10/08/10	1,619.86	Mn
EXPA-000040-V	Matiporon Mining Corp.	11/03/04	8,545.46	Au, Cu, Ag, etc.
EXPA-000041-V	Pearl Asian Mining Industries Inc.	11/24/04	1,127.59	Au, Cu, etc.
EXPA-000044-V	Pearl Asian Mining Industries Inc.	02/17/05	679.88	Au, etc.
EXPA-000113-V	Minevault Inc.	12/20/07	7,723.07	Cu, Au, etc.
EXPA-000114-V	Geo8 Resources Inc.	12/20/07	11,755.43	Cu, Au, etc.

Table 23.1 *Mineral tenements adjacent to MGP. Data sourced from DENR, May 2012.*

Details of mining and exploration activities (where they exist) are detailed below.

Vulcan Industrial & Mining Corporation (VIMC) is a Philippines based company engaged in exploration, development and production of oil and gas reserves, metallic mineral deposits and rock aggregate. VIMC hold tenements on the lower reaches of both the Lanang and Guinobatan Rivers. CGA could not find any relevant descriptions of these properties available in the public domain.

Geograce Resources Philippines Inc. (Geograce) in June 2008 formed a joint venture with Vale Exploration Philippines Inc. (Vale) to cooperate in the exploration of seven tenements located in Masbate and Ticao Islands and which total an area of 84,046 ha. At the same time, Geograce entered into agreements with the current owners of the seven tenements to act on their behalf in conjunction with Vale and an option to purchase all shares in the current owners under terms and conditions which they may mutually agree on at a future date. Three of the seven tenements involved are located adjacent or close by the MGP; Minevault, Geo8 Resources and Masbate 10.

Geograce state that the Masbate tenements "have copper, lead, zinc, arsenic, silver, mercury, pH and electrical conductivity anomalies that coincide with known areas prospective for epithermal gold and copper deposits and show some additional anomalies that have yet to be explored". Geograce, in consultation with Vale, conducted exploration including soil sampling and alteration mapping within the area of Masbate 10's tenement EP-V-2007-004 during 2008. On their website, Geograce also make reference to an upcoming aeromagnetic survey. CGA could find no reference to the results of sampling and aerial survey programmes in the public domain.

Pearl Asian Mining Industries (PAIM) tenements were involved in civil court proceedings when the US Securities and Exchange Commission (SEC) filed a complaint against ZNext Mining Corporation, Inc. (formerly PAIM) and its principal, Elvira Gamboa, in the United States District Court during 2009. The SEC allege that from 2004 to 2008, Gamboa caused PAIM to issue numerous materially false and misleading press releases and other disclosures, which gave the impression PAIM owned and operated a viable gold mining operation in the Philippines. In reality, the SEC alleges PAIM is and was nothing but a shell used by Gamboa to generate stock sale profits.

CGA could find no other information in the public domain regarding other mineral tenements or companies listed in Table 23.1.

23.2 Authors Opinion

The qualified person has been unable to verify the information associated with adjacent mineral tenements and the information is not necessarily indicative of mineralisation on the MGP.

24 OTHER RELEVANT DATA AND INFORMATION

24.1 Operating Project

The MGP has been in commercial operation since July 1st 2009. Operating Statistics for the 2010 and 2011 financial years are presented in Table 24.1.

	2010 FY	2011 FY
High Grade Ore Mined (t)	4,004,613	6,017,187
Grade (g/t)	1.36	1.14
Total Mining (BCM)	7,563,311	8,681,373
Milled (t)	4,486,408	6,152,561
Head Grade (g/t)	1.26	1.13
Recovery (%)	83.29	84.37
Gold Production (oz)	150,141	190,053
Cash Cost (USD / oz)	549	638
Realised Price (USD / oz)	1,052	1,213
Cash Margin (USD / oz)	503	575

Table 24.1 *Selected MGP operating statistics, 2010 and 2011 financial years.*

24.2 Small Scale Mining

Illegal, small scale miners (SSM) have been operating within the tenement areas of the MGP since 1990. Atlas attempted to regulate the activity of small scale miners by purchasing ore from registered SSM Cooperatives. With the closure of Atlas mining operations in 1994, small scale mining numbers and activities were uncontrolled. Several separate groups are involved in SSM activity and includes local and migrant miners using predominantly hand held mining methods who sell ore to a number of small processing plants. Ore is often first processed through rod mills with gold recovered from crushed rock using mercury to form amalgams. As recovery using this method is low (typically < 20%), rod mill tailings are often on-sold to processing plants that utilise cyanide in leach tanks for further gold recovery. Tailings disposal is not regulated and there is evidence that SSM activity is having a detrimental effect on the local environment (section 20.3).

In a survey conducted on site during 2004, 2,700 individuals were directly engaged in small scale mining and processing operations in the vicinity of the MGP, the majority engaging in the activities within FRC property (SKM 2005). SSM numbers have reduced since the commencement of open pit mining operations at the MGP.

Although it is not liable for contamination from SSM activity, CGA has been diligent in attempting to limit the activities of SSM and informing the public about the risk of contamination. In line with attempts to limit and control SSM activity CGA, in coordination with local and National government is endeavouring to enter into agreements with small scale miners. The agreements will form local cooperatives to legally work on some areas of MGP mineral tenements outside of current operations that are not suitable for large scale mining.

24.3 Seismicity

As noted in Section 7, the MGP is located in close proximity to the Philippine Fault Zone (PFZ), a major tectonic feature that transects the whole Philippine archipelago and is seismically active.

The local expression of the PFZ is known as the Masbate Fault which links to the Guinyangan Fault to the north and Central Leyte Fault to the south. Historical earthquakes along the PFZ in the region of Masbate are illustrated in Figure 24.1. Analysis of historical records and detailed analysis of recent seismic events suggest the Masbate Fault is a transition zone characterised by medium to large (Magnitude 5 – 7) earthquakes accompanied by unusually large ground ruptures with post-seismic deformation and possible slow-slip components (Besana et al. 2005).

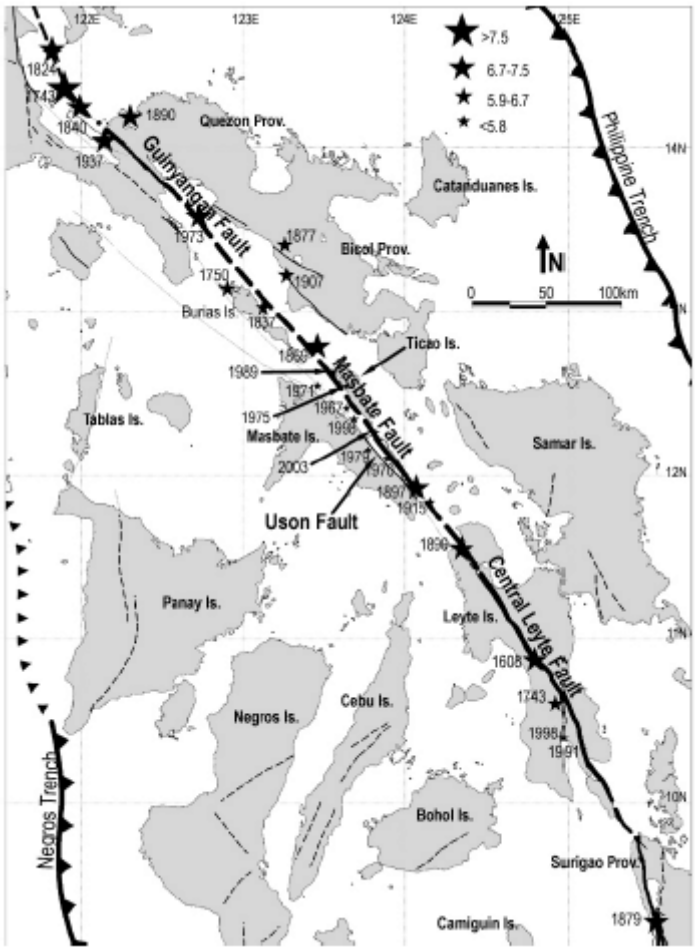


Figure 24.1 *Historical earthquakes, central section of the Philippine Fault Zone (after Besana et al. 2005).*

The most recent seismic event in the Masbate region occurred on 5th March 2012 with a magnitude 5.6 earthquake located at depth of 37 km occurred near Masbate City. Minor damage to buildings and a small number of personal injuries were recorded in Masbate City. Ground movement associated with the earthquake was felt at the MGP but there was no damage to any equipment or facilities.

Construction of critical facilities of the MGP (process plant, power station, TSF) incorporate design criteria matched to the local seismic risk.

25 INTERPRETATION AND CONCLUSIONS

25.1 Summary

MGP is an operating open pit, CIL processing gold operation located in the municipality of Aroroy, Masbate Island, Region V, Republic of the Philippines. Contiguous mineral tenements held, or under application by the project total 11,593 ha.

MGP is owned and operated by three separate Philippine registered companies: FRC, VMC and PGPRC. FRC and VMC hold mineral tenements and operating permits, FRC manages open pit mining operations, while PGPRC processes the ore sold to it by FRC and VMC. CGA indirectly owns 100 % of PGPRC, and 64 % of FRC and VMC.

Gold mining activity has been conducted within the MGP area since at least the 17th Century. Previous owners of the project operated an open pit / underground mining operation and associated CIL processing plant during the period 1980 to 1994 to produce over 1 M oz of gold from processing of 17 Mt of ore. The current mining operation declared commercial operation on 1st July 2009 following construction of the processing facility.

Local geology consists of a series of volcanic rocks of andesitic composition and clastic sediments, predominantly conglomerate. Gold mineralisation is hosted in a series of quartz reefs developed in a network of fault structures and deposited from hydrothermal fluids. Renewed movement along fault structure post mineralisation has deformed many of the quartz reefs and allowed the passage of low temperature fluids resulting in an overprinting of the original alteration assemblage. The MGP is classified as a low sulphidation epithermal deposit.

A series of diamond and RC drilling campaigns of mineralised deposits has been conducted at the MGP under the direction of successive owners from 1975 to the present. The most recent drilling campaign, directed by CGA, commenced in late 2009 and is ongoing. Drilling, sampling and assay data is of sufficient quality to allow estimation of mineral resources.

Data from drilling over the period 1975 to mid-2011 has been used by Mining Associates (MA) to complete a MGP mineral resource estimate as of 31st October 2011. The estimation process involved the construction of numerous vein and halo wireframes that accurately defined the shape and boundaries of mineralisation. Ordinary kriging was utilised to estimate gold grade based on drill hole assay information. The resulting block model incorporates bulk density information based on the degree of weathering and accounts for the presence of historical underground workings. Resource classification was assigned based on estimation parameters. Table 25.1 shows the resulting October 2011 MGP mineral resource estimate at a cut-off grade of 0.36 g/t Au.

Resource Category	Tonnage (Mt)	Au Grade (g/t)	Au Ounces (Moz.)
Measured	6.4	0.77	0.16
Indicated	197.8	0.76	4.84
Total Measured + Indicated	***204.3***	***0.76***	***5.00***
Inferred	102.8	0.86	2.83

Table 25.1 *October 2011 MGP combined mineral resource estimate, 0.36 g/t cut off.*

The new mineral resource estimate is an update to the May 2008 work, having incorporated new

drilled results conducted since 2008 plus reconciliation with the mine production up to October 2011. The new model shows a good correlation over this period with reconciled grade control results, supporting the drop in overall grade noted both between the 2008 to 2011 estimates.

Table 25.2 shows a comparison between the new model and the 2008 model at a cut-off grade of 0.36 g/t Au. The 2011 estimate contains slightly more total resource ounces (7.83 M versus 7.77 M) and replaces the depletion due to mining since operations commenced in 2009.

Category	2011 Resource estimate			2008 Resource estimate		
	Mt	***g/t***	***Moz.***	***Mt***	***g/t***	***Moz.***
Measured	6.4	0.77	0.16			
Indicated	197.8	0.76	4.84	153.4	0.92	4.55
Total M + I	***204.3***	***0.76***	***5.00***	***153.4***	***0.92***	***4.55***
Inferred	102.8	0.86	2.83	127.2	0.79	3.22

Table 25.2 *Comparison of 2011 and 2008 MGP combined mineral resource estimates, 0.36 g/t Au cut-off.*

The 2011 mineral resource estimate has been utilised by Golder Associates to generate an MGP mineral reserve estimate as of 31st October 2011. Resources have been converted to reserves using a cost structure based on current operations, metallurgical recoveries from current processing, pit wall angles derived from independent geotechnical analysis and a gold price of USD 1,300 per oz. Reserves include a mining dilution of 5 %, with diluting material assigned a grade of 0.17 g/t, and a mining recovery of 97.5 %. Final pit designs completed by Golder consider the location of historical underground mining and geological faults.

Ore with a grade between 0.40 to 0.70 g/t Au stockpiled by FRC from commencement of mining in 2009 to 31st October 2011 is included within the reserve estimate. Table 25.3 shows the resulting October 2011 MGP mineral reserve estimate.

Resource Category	Tonnage (Mt)	Au Grade (g/t)	Au Ounces (Moz.)
Proven	5.7	0.75	0.139
Probable	105.5	0.85	2.899
Stockpiles (Probable)	7.4	0.55	0.131
Total P + P	118.6	0.83	3.168

Table 25.3 *October 2011 MGP combined mineral reserve estimate. Cut-off values; oxide material = 0.36 g/t; transitional material 0.39 g/t; primary material 0.43 g/t.*

The 2011 MGP mineral reserve estimate replaces a previous estimate completed in October 2008 (Table 25.4). The 2011 estimate contains more total reserve ounces than the previous estimate in 2008 (3.168 M versus 3.03 M) and replaces the depletion due to mining since operations commenced in 2009. Pit shells generated during both the 2011 and 2008 reserve estimates have similar footprints. The increase seen in the 2011 reserve tonnage includes an element of conversion of Inferred resources from the 2008 mineral resource estimate to higher classifications.

Category	2011 Reserve estimate			2008 Reserve estimate		
	Mt	***g/t***	***Moz.***	***Mt***	***g/t***	***Moz.***
Proven	5.7	0.75	0.139			
Probable	105.5	0.85	2.899	92.2	1.0	3.03
Stockpiles	7.4	0.55	0.131			
Total	**118.6**	**0.83**	**3.168**	**92.2**	**1.0**	**3.03**

Table 25.4 *Comparison of 2011 and 2008 MGP mineral reserve estimates.*

MGP declared commercial production on 1st July 2009. From the date of commercial production until 31st March 2012 MGP has produced over 432,000 oz. Au at an adjusted cash cost of USD 653 / oz. from processing of 13.6 Mt of ore (Table 25.5).

	2010 FY	2011 FY	2012 FY*	PTD
High Grade Ore Mined (t)	4,004,613	6,017,187	3,330,611	13,342,411
Grade (g/t)	1.36	1.14	1.05	1.18
Total Mining (BCM)	7,563,311	8,681,373	5,015,976	21,260,660
Milled (t)	4,486,408	6,152,561	3,019,627	13,658,596
Head Grade (g/t)	1.26	1.13	1.08	1.16
Recovery (%)	83.29	84.37	88.15	84.85
Gold Production (oz)	150,141	190,053	91,157	432,151
Cash Cost (USD / oz)	549	638	844	653
Cash Cost (USD / t milled)	18.35	19.86	26.24	20.71

Table 25.5 *Selected operating statistics, MGP. * = July 1 2011 to March 31 2012. PTD = Project to date. Figures in 2012 FY* reflect reduced production due to SAG Mill repair during the period July 2011 to December 2011.*

Open pit mining is conducted by Leighton Asia Limited (Leighton) under an alliance contract with FRC and PGPRC. Mining uses backhoe excavators and off highway haul trucks operating on 2.5 m bench heights. Ore is defined using dedicated RC grade control drill rigs. Current mining movements are approximately 850,000 to 900,000 BCM per month.

Processing is conducted using a conventional SABC grinding circuit and CIL. Since commissioning, mill throughput on an annualised basis has increased from 4 Mtpa to 6.5 Mtpa. The expenditure of USD 12M between September 2010 and September 2011 resulted in the ability to maintain 6.5 Mtpa throughput on blends with 65 % primary ore content. Use of lead nitrate as a reaction catalyst has maintained or improved metallurgical recovery, despite reduced residence time within the CIL circuit at the higher throughput rates. Process tailings are deposited in an engineered tailings storage facility (TSF) located 2.5 km from the Masbate Process plant.

MGP works closely with government bodies and local stakeholders to maintain all necessary permits, certificates and licences in good standing.

MGP tenements include known gold mineralisation outside of the current mineral resource and mineral reserve estimates. CGA have initiated a significant programme with annual budget of USD 20M to explore the tenement areas in addition to ongoing resource definition of near mine targets. In the opinion of the qualified persons, both exploration and continued resource definition have the potential to increase resources and reserves in future estimates.

25.2 Risks

There is a low level of risk associated with the 2011 mineral resource and mineral reserve estimates. Current drilling, sampling and assay practices are in line with industry standards and historical drilling data has been independently verified by several previous consultants. Estimates of tonnage and grade are in line with ore defined by grade control during mining. Conversion of resources to reserves utilise inputs (costs, metallurgical recoveries, pit wall angles, etc.) that are consistent with current mining and processing of the ore body.

Events that pose significant risk to the continued economic performance has been analysed using the principles and guidelines of risk management (ISO 31000:2009). A risk matrix approach has been taken (Appendix 2). Results of project risk analysis are presented in Table 25.6.

Description of Risk	Likelihood	Severity	Rating	Mitigation measures
Loss of operating Licence	Rare	Catastrophic	High	Open dialogue with stakeholders
Breach of TSF	Rare	Catastrophic	High	Design, freeboard, inspection
Depressed gold price	Possible	Moderate	High	
Major equipment failure (Plant)	Possible	Moderate	High	Redundancy, inspection
Pit wall collapse	Unlikely	Moderate	Medium	
Damage from major earthquake	Rare	Moderate	Medium	
Pit flooding from typhoon	Possible	Minor	Medium	
Civil Unrest	Possible	Insignificant	Low	
Cost Inflation	Likely	Insignificant	Low	

Table 25.6 *Summary of project risks, MGP.*

Mitigation measures for project risks of high ranking are included in Table 25.6.

The MGP has worked hard to build strong relationships with all regulatory and community stakeholders, and is fully compliant with the numerous permits, certificates and licences required to operate a large scale mining operation in the Philippines.

Structures, including the TSF, incorporate local seismic and climatic conditions into design and construction. The TSF is designed and constructed in line with industry practice and subjected to regular inspection and reporting by FRC, DENR and independent bodies.

Some redundancy is available within the process plant to maintain production despite the loss of critical equipment. Significant expenditure over the last 12 months of operation has significantly increased store line items and critical spares. A new SAG mill rolling element is on order with expected delivery of early 2013. Inspection and maintenance of plant equipment is in line with standard industry practice.

25.3 Conclusions

The MGP is an economically successful, modern gold mining operation. 2011 updated mineral resource and mineral reserve estimates represent an increase above previous (2008) estimates and replace reserves depleted by mining and processing over the 27 month period between 1st July 2009 and 31st October 2011.

26 RECOMMENDATIONS

26.1 Exploration

FRC and VMC hold tenements located to both the north and south of current mining areas. Current tenement holdings are prospective in terms of identifying additional areas of epithermal Au mineralisation and have potential for discovery of porphyry style Cu and / or Au mineralisation. A continued focus on local and regional exploration is recommended.

CGA have allocated a significant annual budget (USD 20M) towards local and regional exploration. Exploration activities planned within the next 12 month period include:

- continued drilling of Pajo deposit to allow calculation of mineral resource
- drilling of Au targets identified by previous work (Water West, Young Lady, David Sum, Bart-Ag)
- drilling testing of outcropping Cu-mineralisation at a location near Baleno
- completion of regional field mapping and surface sampling within tenement areas with subsequent analysis and prioritising of further work within anomalous areas
- IP surveys over prospective mineralisation

Exploration success is not critical to current mining activity within the MGP. Exploration success however has the potential to extend the life and improve the economic performance of the current operations. Discovery of porphyry style mineralisation has the potential to significantly increase the value of MGP tenements.

26.2 Resource to Reserve conversion

Inferred category resources included within pit shells of the 2011 mineral reserve estimate total 12 Mt @ 1.00 g/t Au, representing 0.385 Moz of potential reserves if more confidence could be placed on its location and grade. Additional drilling within or immediately adjacent to mineral reserve pit shells to convert Inferred category resources to higher classification is recommended.

CGA have included resource definition drilling within its budget allocation to exploration. Resource definition activities planned within the next 12 month period include:

- Additional drilling to upgrade resource classification in the Blue Quartz and Old Lady deposits
- Additional drilling within Libra, Main Vein, Holy Moses – Basalt, Panique and Colorado deposits with the aim of extending the along strike and down dip definition of known mineralisation, in addition to increasing drill density in areas currently classified as Inferred.
- Extending drill coverage of the Montana deposit to the north west

Further conversion of mineral resources to mineral reserves is not critical to current mining activity within the MGP. Successful conversion however, has the potential to extend mine life and improve economic returns from current MGP operations.

26.3 Increased Production

The current MGP operates under a "bulk tonnage, low grade" scenario. Given updated mineral reserve estimates (113 Mt) there is potential to increase economic returns by increasing annual process throughputs above current levels (6.5 Mtpa). It is recommended that CGA conduct an economic assessment of increased throughput of the MGP. Analysis should address:

- Capital cost estimates associated with process plant upgrade
- Impact to operating unit costs resulting from increased annual throughput
- Identification of the optimal magnitude of throughput increase given capital input required
- Plant and equipment requirements required for increased throughput and impact to the current Masbate Process plant
- Preliminary assessment of increased throughput to related infrastructure (power station, port, reagent handling and storage facilities, TSF, etc.)

Costs estimated to complete an economic assessment are USD 0.5M.

27 CERTIFICATES OF QUALIFIED PERSONS

27.1 Mark Turner

I, Mark B Turner, hereby certify that:

I am a Mining Engineer, employed by CGA Mining Limited, residing at 101 Fairway Circle Connolly Western Australia 6027, Australia with my office at Level 5 BGC Centre , 28 The Esplanade, Perth 6000, Australia (Telephone +61-8-9263-4000).

I graduated from the University of New South Wales, Sydney, Australia in 1985 with a Bachelor Degree of Engineering in the field of Mining Engineering with Honours. I have continuously practised my profession as a Mining Engineer since graduation, in the fields of Mining and Management. I have held technical positions with Conzinc Riotinto of Australia Ltd ("CRA") (now Rio Tinto Limited) and Newcrest Mining Limited and senior positions with Resolute Mining Ltd and CGA Mining Limited.

I have prepared in depth reviews and/or reserve estimates of a number of deposits over the last 18 years, including the Hampton Boulder Deposit, Marymia Open Pit Deposits, Higginsville Open Pits, Chalice Gold Deposit, Obotan Deposit and Golden Pride Open Pit and the Masbate Open Pits.

I have been a Member of the Australasian Institute of Mining and Metallurgy ("The AusIMM") since 1989 and I am also a Certified Professional (Management).

Since being employed by CGA Limited in October 2008, I have made regular (more than 1 per month) visits to the Masbate mine site and have been intimately involved in the mine design, scheduling and optimisation of the project.

I am responsible for all Sections of this Technical Report except Sections 12, 14 and 15.

For the purposes of the Technical Report entitled "NI43-101 Technical Report, Masbate Gold Project, Republic of the Philippines, October 2011" with an effective date of October 31 2011, of which I am an author, I am a Qualified Person as defined in National Instrument 43-101 ("the Policy") and a competent Person as defined under the JORC code (2004). I have read the Policy and this report is prepared in compliance with its provisions. I have read the definition of "qualified person" set out in the Policy and certify that by reason of my education, affiliation with a profession association (as defined in the Policy) and past relevant experience, I fulfil the requirement to be a "qualified person" for the purposes of the Policy.

As of the effective date of the report and to the best of my knowledge, information and belief, the report contains all scientific and technical information that is required to be disclosed in order to make this report not misleading.

I, as a qualified person, am not independent of CGA Mining Limited applying the test set out in Section 1.5 of the Policy as I am a full-time employee of CGA Mining Limited and hold securities of CGA Mining Limited.

I have had no involvement with the property, which is the subject of this report, prior to my current employment with CGA Mining Limited.

Dated at Perth this 20th day of June 2012.

Respectfully submitted

Mark Barlow Turner
B Eng. (Mining)(Hons) MAusIMM CP(Man)
Qualified Person

27.2 Andrew J Vigar

I, Andrew J Vigar, hereby certify that:

I am an independent Consulting Geologist and Professional Geoscientist residing at 97 Isaac Street, Spring Hill Queensland 4000, Australia with my office at Level 4, 67 St Paul's Terrace, Brisbane, Queensland 4001, Australia (Telephone +61-7-3831-9154).

I graduated from the Queensland University of Technology, Brisbane, Australia in 1978 with a Bachelor Degree in Applied Science in the field of Geology. I have continuously practised my profession as a Geologist since graduation, in the fields of mineral exploration, mine geology and mineral resource estimation. I have held senior positions with Emperor Gold, Western Mining Corporation, Costain Australia and Conzinc Riotinto of Australia Ltd ("CRA") (now Rio Tinto Limited) prior to commencing full-time consulting in 1996. I have been involved in consulting to the minerals industry both independently (Vigar & Associates and now Mining Associates Limited, and Mining Associates Limited) and as an employee of the international consultancy, SRK Consulting.

My specific experience concerning the Masbate deposit is my extensive experience in mineral deposits in volcanic terrains, specifically Emperor Gold Mine, an alkalic epithermal deposit. I worked on the Wafi and Hidden Valley gold project in Papua New Guinea with CRAE (now Rio Tinto). I have prepared in-depth reviews and/or resource estimates of a large number of deposits over the last 10 years, including the Masbate Gold Deposit and the Runruno Gold-Molybdenum Deposit 2008 and 2009 respectively. In a more general sense, I have worked on the identification and estimation of resources for epithermal/hydrothermal and porphyry-style mineralisation in similar environments in Papua New Guinea, Fiji, New Zealand, Philippines, Indonesia and Southern China as well as throughout Australia.

I was elected a Fellow of the Australasian Institute of Mining and Metallurgy ("The AusIMM") in 1993, having been a member since 1980. My status as a Fellow of The AusIMM is current, and I am recognized by the Australian Securities and Investments Commission and the Australian Stock Exchange as a Qualified Person for the submission of Independent Geologist's Reports.

I first visited Masbate in 1996 and 1997 and most recently from 5th to the 8th of August 2011.

I am responsible for Sections 1, 10, 11, 12, 14, 25 and 26 of this Technical Report.

For the purposes of the Technical Report entitled: "NI43-101 *Technical Report, Masbate Gold Project, Republic of the Philippines, October 2011*" with an effective date of October 31, 2011, of which I am an author, I am a Qualified Person as defined in National Instrument 43-101 ("the Policy") and a Competent Person as defined under the JORC code (2004).

I have read the Policy and this technical report is prepared in compliance with its provisions. I have read the definition of "qualified person" set out in the Policy and certify that by reason of my education, affiliation with a professional association (as defined in the Policy) and past relevant work experience, I fulfil the requirement to be a "qualified person" for the purposes of the Policy.

As of the effective date of the report and to the best of my knowledge, information and belief, the report contains all scientific and technical information that is required to be disclosed in order to make this report not misleading.

I have no direct or indirect interest in the properties which are the subject of this report. I do not hold, directly or indirectly, any shares in CGA Mining Limited or other companies with interests in the Masbate Gold Project. I am independent of CGA Mining Limited as set out in Section 1.5 of NI 43-101 and Section 3.5 of NI 43-101CP.

I have had no prior involvement with the property which is the subject of this report other than my previous involvement with past NI43-101 reports.

I will receive only normal consulting fees for the preparation of this report.

Dated at Brisbane this 20th day of June 2012.

Respectfully submitted

Andrew James Vigar

B App.Sc.(Geol)
Qualified Person

27.3 Stephen Jones

I, Stephen Jones, hereby certify that:

I am an independent Consulting Mining Engineer residing at 41 Blythewood Road Torrens Park South Australia 5062, Australia with my office at 99 Franklin Street, Adelaide, South Australia 5000, Australia (Telephone +61-8-8235-2100).

I graduated from the South Australian Institute of Technology (now University of South Australia), Adelaide, Australia in 1986 with a Bachelor Degree in Engineering in the field of Mining Engineering. I have continuously practised my profession as a Mining Engineer since graduation, in the fields of Mining and Management. I have held senior positions with Resolute Mining, Sons of Gwalia, Brandrill, Consolidated Minerals, Kagara Ltd and Golder. I have been involved in consulting to the minerals industry as an employee of the international consultancy, Golder.

I have prepared in-depth reviews and/or reserve estimates of a number of deposits over the last 10 years, including the Chalice Gold Deposit, the Mt Garnet Zinc open cut and underground and Mungana Copper/Gold Open Pit and underground.

I was elected a Fellow of the Australasian Institute of Mining and Metallurgy ("The AusIMM") in 2011, having been a member since 2000. I am a Certified Professional (Mining) as well as being a Registered Professional Engineer of Queensland (RPEQ 10665).

I visited Masbate recently from 5th to the 8th of November 2011.

I am responsible for Sections 1,15,25,26 of this Technical Report.

For the purposes of the Technical Report entitled: "NI43-101 *Technical Report, Masbate Gold Project, Republic of the Philippines, October 2011*" with an effective date of October 31, 2011, of which I am an author, I am a Qualified Person as defined in National Instrument 43-101 ("the Policy") and a Competent Person as defined under the JORC code (2004).

I have read the Policy and this technical report is prepared in compliance with its provisions. I have read the definition of "qualified person" set out in the Policy and certify that by reason of my education, affiliation with a professional association (as defined in the Policy) and past relevant work experience, I fulfil the requirement to be a "qualified person" for the purposes of the Policy.

As of the effective date of the report and to the best of my knowledge, information and belief, the report contains all scientific and technical information that is required to be disclosed in order to make this report not misleading.

I have no direct or indirect interest in the properties which are the subject of this report. I do not hold, directly or indirectly, any shares in CGA Mining Limited or other companies with interests in the Masbate Gold Project. I am independent of CGA Mining Limited as set out in Section 1.5 of NI 43-101 and Section 3.5 of NI 43-101CP.

I have had no prior involvement with the property which is the subject of this report.

I will receive only normal consulting fees for the preparation of this report.

Dated at Adelaide this 20th day of June 2012.

Respectfully submitted

Stephen Thornton Jones
B Eng.(Mining) FAusIMM CP(Min)

Qualified Person

28 CONSENT OF QUALIFIED PERSONS

28.1 Mark Turner

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CGA Mining Limited

CONSENT OF QUALIFIED PERSON

MARK B TURNER

Ontario Securities Commission
Alberta Securities Commission
British Colombia Securities Commission

I, Mark B Turner, B Eng. (Mining) (Hons), MAusIMM, CP (Man), do hereby consent to the public filing of the technical report titled "NI43-101 Technical Report, Masbate Gold Project, Republic of the Philippines October 2011" (the "Technical Report"), prepared for CGA Mining Limited and dated 31 October 2011.

I also consent to the use of any extracts from, or a summary of, the Technical Report in the press release dated 07 May 2012 of CGA Mining Limited (the "Press Release").

I certify that I have read the Press Release and that it fairly and accurately represents the information in the sections of the Technical Report for which I am responsible.

Dated the 20th day of June, 2012.

Mark B Turner, B Eng. (Mining) (Hons), MAusIMM, CP (Man)

28.2 Andrew Vigar

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CONSENT OF QUALIFIED PERSON

ANDREW J VIGAR

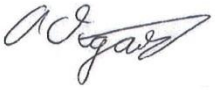
Ontario Securities Commission
Alberta Securities Commission
British Colombia Securities Commission

I, Andrew J. Vigar, BAppSc Geo, FAusIMM, MSEG, do hereby consent to the public filing of the technical report titled "NI43-101 Technical Report, Masbate Gold Project, Republic of the Philippines, October 2011" (the "Technical Report"), prepared for CGA Mining Limited and dated 31 October 2011.

I also consent to the use of any extracts from, or a summary of, the Technical Report in the press release dated 07 May 2012 of CGA Mining Limited (the "Press Release").

I certify that I have read the Press Release and that it fairly and accurately represents the information in the sections of the Technical Report for which I am responsible.

Dated the 20 day of June, 2012.



Andrew J. Vigar, BAppSc Geo, FAusIMM, MSEG

28.3 Stephen Jones

Golder Associates

CONSENT OF QUALIFIED PERSON

STEPHEN JONES

Ontario Securities Commission
Alberta Securities Commission
British Colombia Securities Commission

I, Stephen Jones, BEng Mining, FAusIMM, (CP), do hereby consent to the public filing of the technical report titled "NI43-101 Technical Report, Masbate Gold Project, Republic of the Philippines, October 2011" (the "Technical Report"), prepared for CGA Mining Limited and dated 31 October 2011.

I also consent to the use of any extracts from, or a summary of, the Technical Report in the press release dated 07 May 2012 of CGA Mining Limited (the "Press Release").

I certify that I have read the Press Release and that it fairly and accurately represents the information in the sections of the Technical Report for which I am responsible.

Dated the 20th day of June 2012.

Stephen Jones, BEng Mining, FAusIMM, (CP)

BRW. CLIENT CHOICE AWARDS 2012 WINNER

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29 REFERENCES

29.1 Abbreviations

AAS	Atomic absorption spectroscopy
AEPEP	Annual Environmental Protection & Enhancement Programme
Ai	Abrasion Index
Ag	silver
APSA	Application for Production Sharing Agreement
ASX	Australian stock exchange
Au	gold
BCM	banked cubic metre
BDA	Behre Dolbear Australia
BWi	Bond Work Index
CAAP	Civil Aviation Authority of the Philippines
CGA	CGA Mining Limited
CIL	Carbon in Leach
CIP	Carbon in Pulp
CLRF	Contingent Liabilities & Rehabilitation Fund
cm	centimetre
Co	Company
CSR	Corporate social responsibility
Cu	copper
DDH	diamond drill hole
DENR	Department for the Environment and Natural Resources
DH	drill hole
E	east
ECC	Environmental Compliance Certificate
EDCO	Exploration Drilling Corporation
EFT	Environment Trust Fund
EIS	Environmental Impact Statement
EMB	Environmental Management Board
EOH	end of hole
EP	Exploration Permit
EPEP	Environmental Protection & Enhancement Programme
ESE	east southeast
EWP	Environmental work programme
EXPA	Application for Exploration Permit
FMRDF	Final Mine Rehabilitation & Decommissioning Fund
FRC	Filminera Resources Corporation
FY	Financial Year (1 July to 30 June)
g	grams
g/t	grams per tonne
ha	hectare
HFO	heavy fuel oil
HQ	diamond drill tube size – 63.5 mm inside diameter
ICP	Inductively coupled plasma
ICPAES	Inductively coupled plasma – atomic emission spectrometry
ID	identity
IMC	IMC Consultants Pty. Limited
IP	Indigenous Peoples
IP	Induced Polarisation
ISO	International Organisation for Standardisation
kg	kilogram
km	kilometre
kt	thousand tonnes
koz	thousand troy ounces

kWh	kilowatt hour
m	metre
M	million
m^3	cubic metre
Ma	mega annum (unit of time equivalent to one million years)
MA	Mining Associates Pty. Limited
MGB	Mines & Geosciences Board
MGO	Masbate Gold Operations
MGP	Masbate Gold Project
mm	millimetre
MMT	Multipartite Monitoring Team
MPP	Mineral Processing Permit
MPSA	Mineral Production Sharing Agreement
Moz	million troy ounces
MRF	Mine Rehabilitation Fund
mRL	metres above sea level
MSPU	mobile sample preparation unit
Mt	million tonnes
Mt	Mount
MTF	Monitoring Trust Fund
Mtpa	million tonnes per annum
MW	megawatt
MWTF	Mine waste & tailings fees
N	North
NQ	diamond drill tube size – 47.6 mm inside diameter
nT	nano Tesla
NW	northwest
NWRB	National Water Resources Board
OK	Ordinary Kriging
oz	Troy ounce (= 31.103477 grams)
PAIM	Pearl Asian Mining Industries
PFZ	Philippine Fault Zone
PGO	Philippines Gold Plc.
PGPRC	Philippine Gold Processing & Refining Corporation
PHP	Philippine Peso
PPA	Philippines Port Authority
ppb	part per billion
ppm	part per million
PQ	diamond drill tube size – 85 mm inside diameter
PTD	project to date
P_{80}	80% passing
QAQC	quality assurance / quality control
RAB	Rotary air blast
RC	reverse circulation
RCDH	reverse circulation drill hole
RCF	Rehabilitation Cash Fund
RL	relative level (normally a reference to sea level)
RWi	Bond Rod Work Index
SAG	Semi-autogenous grinding
SDMP	Social Development & Management Plan
SE	southeast
SEC	United States Securities & Exchange Commission
SG	specific gravity
SMU	smallest mining unit
SSM	small scale mining (miners)
SW	southwest
t	metric tonne (= 1,000 kilograms)

t/m^3	tonnes per cubic metre
TSF	Tailings storage facility
TSP	Total suspended particles
TSX	Toronto stock exchange
t:t	metric tonne for metric tonne
USD	United States dollars
VIMC	Vulcan Industrial & Mining Corporation
VMC	Vicar Mining Corporation
VN	vein
WNW	west northwest
wt.	weight
3D	three dimensional
@	at
%	percent
°	degrees
°C	degrees Celsius

29.2 Glossary of Technical Terms

This glossary comprises a general list of common technical terms that are typically used by geologists. The list has been edited to conform in general to actual usage in the body of this report. However, the inclusion of a technical term in this glossary does not necessarily mean that it appears in the body of this report, and no imputation should be drawn. Investors should refer to more comprehensive dictionaries of geology in printed form or available in the internet for a complete glossary.

block model	A block model is a computer based representation of a deposit in which geological zones are defined and filled with blocks which are assigned estimated values of grade and other attributes. The purpose of the block model (BM) is to associate grades with the volume model. The blocks in the BM are basically cubes with the size defined according to certain parameters.
bulk density	The dry in-situ tonnage factor used to convert volumes to tonnage. Bulk density test work is carried out on site and is relatively comprehensive, although samples of the more friable and broken portions of the mineralized zones are often unable to be measured with any degree of confidence, therefore caution is used when using the data.
cut-off grade	The lowest grade value that is included in a resource statement. Must comply with JORC requirement 19 "*reasonable prospects for eventual economic extraction*" the lowest grade, or quality, of mineralized material that qualifies as economically mineable and available in a given deposit. May be defined on the basis of economic evaluation, or on physical or chemical attributes that define an acceptable product specification.
diamond drilling, diamond core	Rotary drilling technique using diamond set or impregnated bits, to cut a solid, continuous core sample of the rock. The core sample is retrieved to the surface, in a core barrel, by a wire line.
down-hole survey	Drill hole deviation as surveyed down-hole by using a conventional single-shot camera and readings taken at regular depth intervals, usually every 50 metres.
drill-hole database	The drilling, surveying, geological and analyses database is produced by qualified personnel and is compiled, validated and maintained in digital and hardcopy formats.
grade cap, also called top cut	The maximum value assigned to individual informing sample composites to reduce bias in the resource estimate. They are capped to prevent over estimation of the total resource as they exert an undue statistical weight. Capped samples may represent "outliers" or a small high-grade portion that is volumetrically too small to be separately domained.
inverse distance estimation	It asserts that samples closer to the point of estimation are more likely to be similar to the sample at the estimation point than samples further away. Samples closer to the point of estimation are collected and weighted according to the inverse of their separation from the point of estimation, so samples closer to the point of estimation receive a higher weight than samples further away. The inverse distance weights can also be raised to a power, generally 2 (also called inverse distance squared). The higher the power, the more weight is assigned to the closer value. A power of 2 was used in the estimate used for comparison with the OK estimates.
Indicated Resource	That part of a Mineral Resource for which tonnage, densities, shape, physical characteristics, grade and mineral content can be estimated with a reasonable level of confidence. It is based on exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes. The locations are too widely or inappropriately spaced to confirm geological and/or grade continuity but are spaced closely enough for continuity to be assumed.
Inferred Resource	That part of a Mineral Resource for which tonnage, grade and mineral content can be estimated with a low level of confidence. It is inferred from geological evidence and assumed but not verified geological and/or grade continuity. It is based on information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes which may be limited or of uncertain quality and reliability.
kriging neighbourhood	The methodology for quantitatively assessing the suitability of a kriging neighbourhood involves some simple tests. It has been argued that KNA is a mandatory step in setting up any kriging

analysis, or KNA	estimate. Kriging is commonly described as a "minimum variance estimator" but this is only true when the block size and neighbourhood are properly defined. The objective of KNA is to determine the combination of search neighbourhood and block size that will result in conditional unbiasedness.
Measured Resource	That part of a Mineral Resource for which tonnage, densities, shape, physical characteristics, grade and mineral content can be estimated with a high level of confidence. It is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes. The locations are spaced closely enough to confirm geological and grade continuity.
Mineral Resource	A concentration or occurrence of material of intrinsic economic interest in or on the Earth's crust in such form, quality and quantity that there are reasonable prospects for eventual economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge. Mineral Resources are sub-divided, in order of increasing geological confidence, into Inferred, Indicated and Measured categories when reporting under NI43-101.
Mineral Reserve	Mineral reserves are mineral resources known to be economically feasible for extraction. Mineral Resources are sub-divided, in order of increasing confidence, into Probable and Proven categories when reporting under NI43-101.
nearest neighbour estimation	Nearest Neighbour assigns values to blocks in the model by assigning the values from the nearest sample point to the block attribute of interest.
NI43-101	National Instrument 43-101 is a mineral resource classification scheme used for the public disclosure of information relating to mineral properties in Canada. The Instrument is a codified set of rules and guidelines for reporting and displaying information related to mineral properties owned by, or explored by, companies which report these results on stock exchanges within Canada.
ordinary Kriging estimation, or OK	Kriging is an inverse distance weighting technique where weights are selected via the variogram according to the samples distance and direction from the point of estimation. The weights are not only derived from the distance between samples and the block to be estimated, but also the distance between the samples themselves. This tends to give much lower weights to individual samples in an area where the samples are clustered. OK is known as the "best linear unbiased estimator. The kriging estimates are controlled by the variogram parameters. The variogram model parameters are interpreted from the data while the search parameters are optimised during kriging neighbourhood analysis.
oz	Troy ounce (= 31.103477 grams). Moz = million troy ounces
Probable Reserve	A Probable Ore Reserve is the economically mineable part of Indicated, and in some circumstances, a Measured Mineral Resource. It includes diluting material and allowances for losses which may occur when the material is mined. A Probable Ore Reserve has a lower level of confidence than a Proved Ore Reserve but is of sufficient quality to serve as the basis for decision on the development of deposit.
Proven Reserve	A Proven Ore Reserve is the economically mineable part of a Measured Mineral Resource. It includes diluting materials and allowances for losses which occur when the material is mined. A Proven Ore Reserve represents the highest confidence category of reserve estimate.
QAQC	Quality Assurance/Quality Control. The procedures for sample collection, analysis and storage. Drill samples are despatched to 'certified' independent analytical laboratories for analyses. Blanks, Duplicates and Certified Reference Material samples should be included with each batch of drill samples as part of the Company's QA/QC program.
RC drilling	Reverse Circulation drilling. A method of rotary drilling in which the sample is returned to the surface, using compressed air, inside the inner-tube of the drill-rod. A face-sampling hammer is used to penetrate the rock and provide crushed and pulverised sample to the surface without contamination.
survey	Comprehensive surveying of drill hole positions, topography, and other cadastral features is carried out by the Company's surveyors using 'total station' instruments or DGPS (Differential Global Positioning System) and independently verified on a regular basis. Locations are stored in

	both local drill grid and UTM coordinates.
t	Tonne (= 1 million grams)
variogram	The Variogram (or more accurately the Semi-variogram) is a method of displaying and modelling the difference in grade between two samples separated by a distance h, called the "lag" distance. It provides the mathematical model of variation with distance upon which the Krige estimation method is based.
wireframe	This is created by using triangulation to produce an isometric projection of, for example, a rock type, mineralization envelope or an underground stope. Volumes can be determined directly of each solid.

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30 APPENDICES

30.1 Appendix 1

30.1.1 Rock Solid Data Consultancy – QC Report

ROCKSOLID
DATA CONSULTANCY

FILMINERA RESOURCES CORPORATION

QC REPORT

data management process review and auditing • database integration and optimisation • database design, implementation and maintenance • data compilation • training and mentoring

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Filminera Resources Corporation QC Report Dec 2011

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1 Executive Summary

Filminera Resources Corporation (Filminera) requested Rock Solid Data Consultancy Pty Ltd (Rock Solid Data) to compile a QC Report based on resource holes completed since 2009 and holes with prefix RCGC in preparation for resource modelling.

This QC report is based on the quality control data associated with the 622 RC and diamond resource drill holes completed between Apr 2010 and Sep 2011 and 5168 RCGC drill holes completed between Jun 2008 and Apr 2010. Filminera collected 60,000 resource RC samples, 13,000 resource diamond samples and over 130,000 RCGC samples from these drill holes. Available quality control data associated with drill samples include field duplicates, laboratory duplicates and analytical repeats along with certified reference material and blanks. The samples were sent to McPhar Laboratories Manila Philippines (McPhar) and SGS Masbate onsite Laboratory (SGS) for analysis by Fire Assay AAS method.

Overall the results are good and provide a reasonable level of confidence in the data. The key issue identified is a low bias in the SGS gold results for certified standards when compared with the expected value; in all cases the low bias is not present in McPhar gold results for the same standards. The bias is present in a few Geostats standards as well as two of the four recently adopted Rocklabs standards. The main concern with a low bias is the affect it would have on a resource model. The other key issue is StandardID mis-identification which is occurring in even the most recent Rocklabs standards; these significant outliers are impacting on the statistics making it difficult to distinguish between true laboratory bias and bias caused by the affect of outliers.

Blanks submitted to SGS during the RCGC drilling campaign show elevated gold levels, however this might be due to the type of material used as a blank rather than laboratory contamination. If the blank material was not sourced from a contamination free environment, then it is possible that the blank samples arrived at the laboratory already contaminated. Further investigation is required to determine the cause. Blanks analysed by McPhar during both the resource and RCGC drilling campaigns perform well and many of the 3% outliers shown in the chart can be attributed to StandardID mis-identifciation by Filminera rather than laboratory contamination.

Quality control issues identified are summarised in this report and outliers are listed in the spreadsheet FRC_QC Report_20111219.xlsx. Issues with respect to individual samples and analytical results are not discussed here; this information can be found in the relevant Filminera QC monthly reports.

Rock Solid Data recommends continuous quality control monitoring on a daily and monthly basis; any issues identified need to be investigated and addressed, especially issues relating to biases or contamination. Should a laboratory bias be detected the responsible geologist must immediately follow-up with the laboratory to determine the cause and solution. Blanks need to be uniquely identified in the database, particularly grade control blanks versus resource blanks. Certified blank material should be labelled with the StandardID from the supplier and not renamed to "Blank". Standard outliers should be investigated to firstly determine if StandardID mis-identification is the cause, and the database corrected if necessary; outliers can disguise true laboratory biases in statistics, so it is critical to minimise these obvious mistakes through ongoing monitoring and management.

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The results of the statistical analysis are visually presented as follows:

- Standard Control Plot shows the assay results of a particular reference standard over time. The results can be compared to the expected value and 2x and 3x the expected standard deviations, providing a good indication of both precision and accuracy over time.
- Correlation Plot is a simple plot of the value of the original assay against the repeat assay. The plot allows an overall visualisation of precision and bias over selected grade ranges. The plot is present in normal scale and log scale.
- Statistics to support the charts include mean, standard deviation and coefficient of variation. Statistics are based on grade ranges relevant to the resource.
- Particle Size Analysis shows a month-by-month trend of the particle size analysis results of the prepared sample. The plot is a good indicator of the performance of the laboratory's sample preparation techniques.

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2 Standards and Blanks

2.1 Certified Reference Material

Certified reference material (standards) were included with the Masbate drill samples at regular intervals at a rate of 1:40 samples. Standards with expected gold values ranging from 0.26ppm to 6.78ppm were supplied by Geostats (10 Marsh Close, O'Connor, Western Australia www.geostats.com.au) and Rocklabs Limited (161 Neilson Street, Auckland, New Zealand www.rocklabs.com). Geostats standards were used until July 2011 after which Filminera started using Rocklabs standards. The standards provide an indication of the accuracy of the analytical results.

Overall the gold grade of the 4657 standards are acceptable; however low biases are present in the SGS Masbate gold results for several standards; the concern with a bias is the impact it might have on resource calculations. McPhar gold results do not show the same low biases. There are also numerous significant outliers in the dataset which are impacting on the statistics.

A low bias of 5% is apparent in the SGS gold results for standards G302-5, G398-4 and G905-9 analysed between May 2009 and May 2010; this bias is not present in the McPhar gold results for these standards. Figure 2.1-1 compares the performance of McPhar and SGS for the three standards. Low biases are also present in all SGS gold results for Rocklabs standards SJ53 (-3%) and SL51 (-3% with outliers excluded); no McPhar population for these standards exists in order to make a comparion.

Additionally, there are a significant number of outliers impacting on the statistics, many of which are due to standard mis-identification. For example, standards G305-2 and G305-3 show (overall) biases of +22% and -13% respectively; however of the 38 G305-2 standards 9 are labelled G305-3, and 8 of the 38 G305-3 are labelled G305-2. These two standards perform well within expected ranges when you take into consideration the apparent mis-identification. The Rocklabs standards show 10 significant outliers; in each case the "outlier" gold result matches exactly with the expected value of another Rocklabs standard or a blank.

Not all outliers can be attributed to standard mis-identification and may be caused by SampleID mix-up with adjacent drill samples or laboratory error.

The analytical performance of the 16 CRM is illustrated in Figures 2.1-1 to 2.1-16. Significant outliers excluded from the charts are listed in the tables 2.1-2 to 2.1-6; all results outside of 3 standards deviations from expected are listed in the accompanying spreadsheet FRC_QC Report_20111219.xlsx

The summary table below shows the certified expected Au ppm value versus the calculated Mean Au ppm value, as well as the calculated min-max range, standard deviation and bias for the McPhar Laboratory and SGS Masbate Laboratory datasets.

Table 2.1-1 : Standards Summary

Standard	Lab	No. of Samps	Expected Au ppm	Mean Au ppm	Min Au ppm	Max Au ppm	SD	CV	Mean Bias
G302-5	McPHAR	353	1.66	1.70	0.46	3.65	0.20	0.12	2.41
G302-5	SGS_mas	539	1.66	1.56	0.02	3.66	0.21	0.13	-5.91
G398-4	McPHAR	365	0.66	0.68	0.57	2.67	0.13	0.19	2.30
G398-4	SGS_mas	545	0.66	0.64	0.01	3.24	0.19	0.29	-2.41
G905-9	McPHAR	363	1.86	1.87	0.01	4.06	0.28	0.15	0.41
G905-9	SGS_mas	545	1.86	1.78	0.04	8.67	0.33	0.18	-4.45
G303-8	McPHAR	193	0.26	0.26	0.21	1.63	0.09	0.36	-0.45
G303-8	SGS_MAS	6	0.26	0.27	0.24	0.30	0.02	0.07	3.85

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Standard	Lab	No. of Samps	Expected Au ppm	Mean Au ppm	Min Au ppm	Max Au ppm	SD	CV	Mean Bias
G305-2	McPHAR	34	0.32	0.41	0.03	0.75	0.19	0.45	29.46
G305-2	SGS_MAS	4	0.32	0.42	0.31	0.69	0.16	0.40	29.69
G305-3	McPHAR	34	0.72	0.64	0.29	0.83	0.17	0.26	-11.74
G305-3	SGS_MAS	4	0.72	0.49	0.28	0.81	0.24	0.49	-32.29
G307-4	McPHAR	192	1.4	1.42	0.26	1.79	0.14	0.10	1.69
G307-4	SGS_MAS	5	1.4	1.44	1.25	1.64	0.11	0.07	2.47
G399-2	McPHAR	23	1.46	1.45	1.30	1.57	0.06	0.04	-0.67
G399-2	SGS_MAS	14	1.46	1.41	1.28	1.68	0.09	0.06	-3.52
G901-5	McPHAR	192	1.65	1.63	0.25	1.92	0.13	0.08	-1.00
G901-5	SGS_MAS	15	1.65	1.54	1.30	1.78	0.13	0.09	-6.69
G907-2	McPHAR	191	0.89	0.93	0.01	7.08	0.57	0.61	4.68
G907-2	SGS_MAS	4	0.89	1.97	0.75	6.30	2.28	1.15	121.80
G907-3	McPHAR	192	2.88	2.90	0.15	6.63	0.38	0.13	0.71
G907-3	SGS_MAS	5	2.88	2.74	2.53	3.03	0.15	0.06	-4.89
G907-8	McPHAR	191	6.78	6.89	0.01	8.00	0.77	0.11	1.64
G907-8	SGS_MAS	4	6.78	6.56	6.09	6.93	0.25	0.04	-3.32
SF45	McPHAR	18	0.848	0.87	0.82	0.90	0.02	0.03	2.98
SF45	SGS_MAS	151	0.848	0.85	0.02	1.37	0.10	0.12	0.38
SH41	McPHAR	13	1.344	1.41	0.84	1.48	0.17	0.12	4.58
SH41	SGS_MAS	153	1.344	1.32	0.66	1.48	0.08	0.06	-1.68
SJ53	McPHAR	1	2.637	2.85	2.85	2.85			8.08
SJ53	SGS_MAS	157	2.637	2.57	1.32	2.88	0.12	0.05	-2.51
SL51	McPHAR	1	5.909	6.80	6.80	6.80			15.08
SL51	SGS_MAS	152	5.909	5.64	0.01	6.29	0.70	0.12	-4.64

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Figure 2.1-1 : G302-5

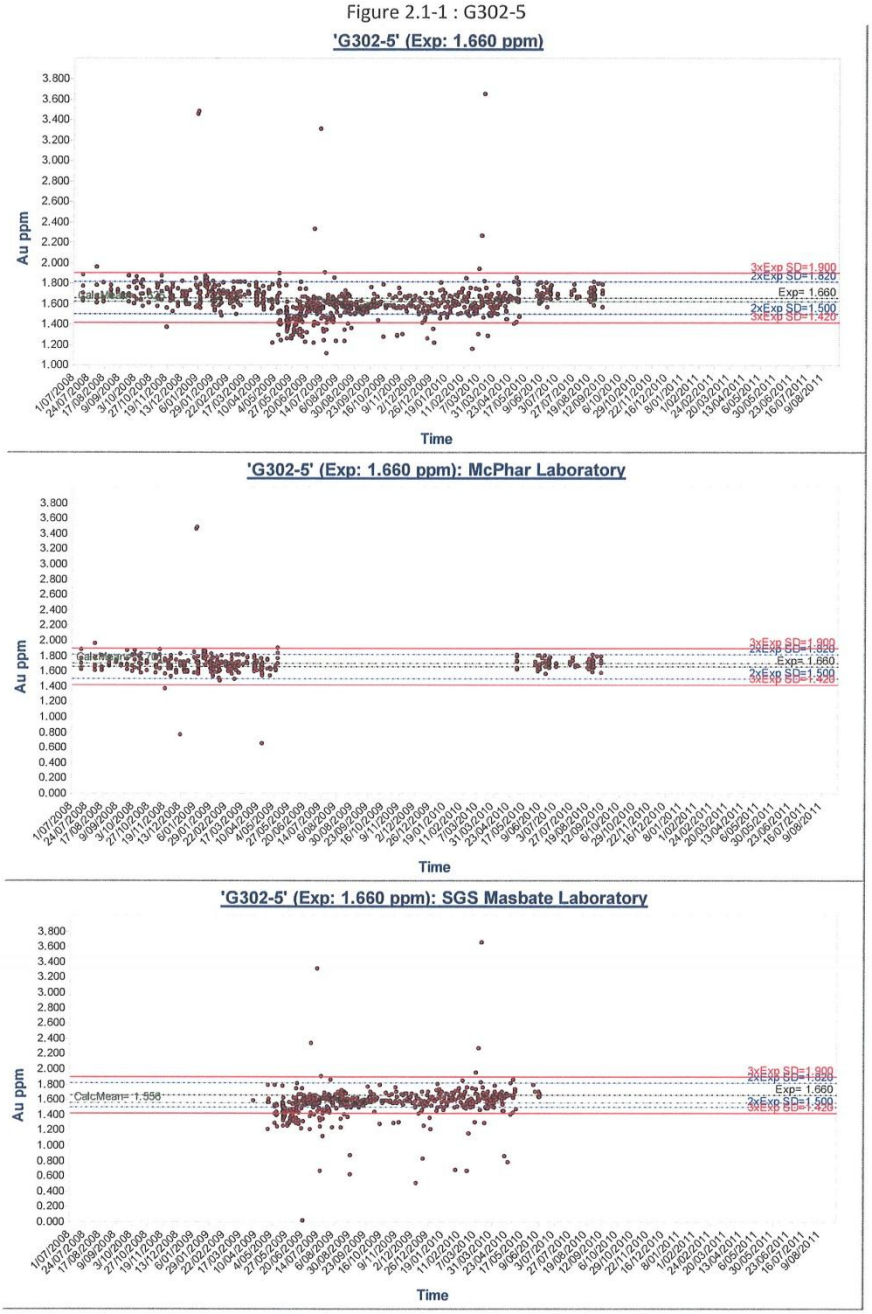

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Figure 2.1-2 : G398-4

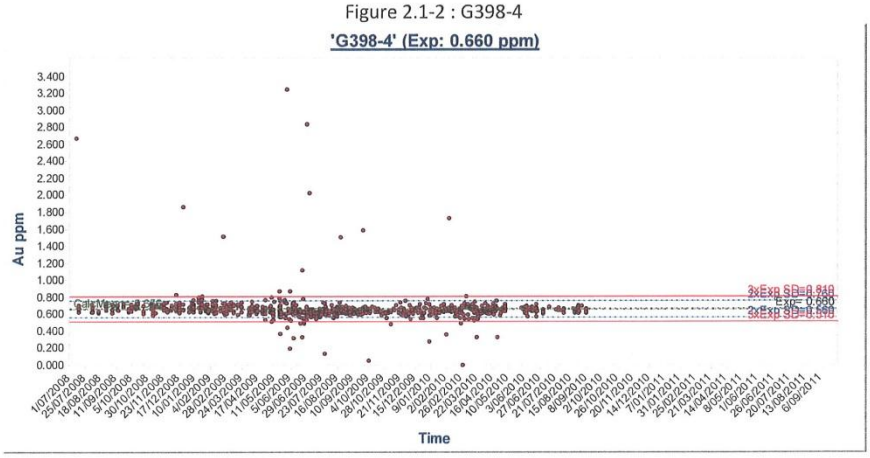

Figure 2.1-3 : G905-9

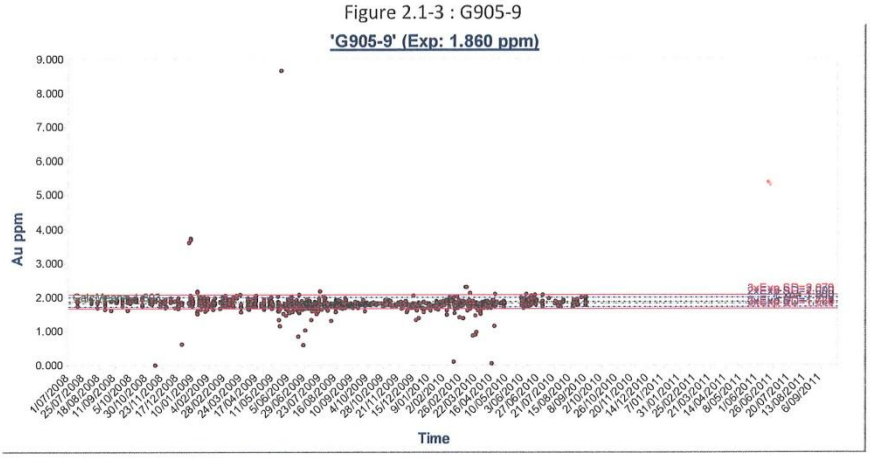

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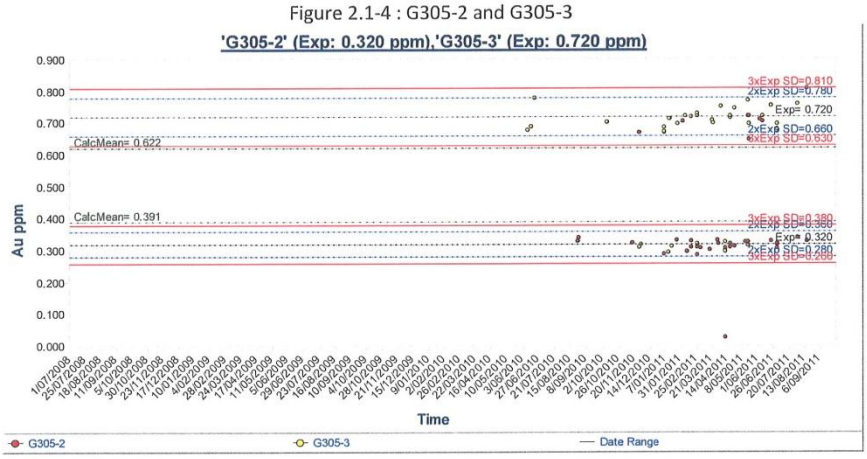
Figure 2.1-4 : G305-2 and G305-3



The name similarity between G305-2 and G305-3 appears to have caused StandardID mis-identification in 22% of the samples. Several G305-2 standards show results within the expected range for G305-3, and vice versa, listed in the table of outliers below.

Table 2.1-2 : Outliers excluded from chart

StandardID	LabCode	Batch_No	Batch_Date	SampleID	Method	Expected	Au ppm	% Diff
G305-2	McPHAR	11-0737	11/04/11	EX634584	FA_AAS	0.32	0.026	92
G305-2	McPHAR	11-0963	18/05/11	EX631644	FA_AAS	0.32	0.649	-103
G305-2	SGS_MAS	MS-4062-10	27/11/10	EX610787	FAA505	0.32	0.67	-109
G305-2	McPHAR	11-1122	08/06/11	EX630510	FA_AAS	0.32	0.706	-121
G305-2	McPHAR	11-0020	03/02/11	EX610987	FA_AAS	0.32	0.707	-121
G305-2	McPHAR	11-1072	04/06/11	EX635286	FA_AAS	0.32	0.712	-123
G305-2	McPHAR	10-2123	14/01/11	EX604298	FA_AAS	0.32	0.715	-123
G305-2	McPHAR	11-0901	17/05/11	EX635783	FA_AAS	0.32	0.722	-126
G305-2	McPHAR	11-0963	18/05/11	EX631742	FA_AAS	0.32	0.724	-126
G305-3	McPHAR	10-2114	11/01/11	EX609781	FA_AAS	0.72	0.294	59
G305-3	McPHAR	11-0737	11/04/11	EX634524	FA_AAS	0.72	0.296	59
G305-3	SGS_MAS	MS-4059-10	27/11/10	EX610722	FAA505	0.72	0.31	57
G305-3	McPHAR	10-2118	17/01/11	EX209310	FA_AAS	0.72	0.314	56
G305-3	McPHAR	11-0901	17/05/11	EX635641	FA_AAS	0.72	0.317	56
G305-3	SGS_MAS	MS-4092-10	29/11/10	EX610918	FAA505	0.72	0.32	56
G305-3	McPHAR	11-0216	25/02/11	EX633483	FA_AAS	0.72	0.322	55
G305-3	McPHAR	11-0737	11/04/11	EX634438	FA_AAS	0.72	0.326	55
G305-3	McPHAR	11-0737	11/04/11	EX634613	FA_AAS	0.72	0.327	55

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Figure 2.1-5 : G303-8

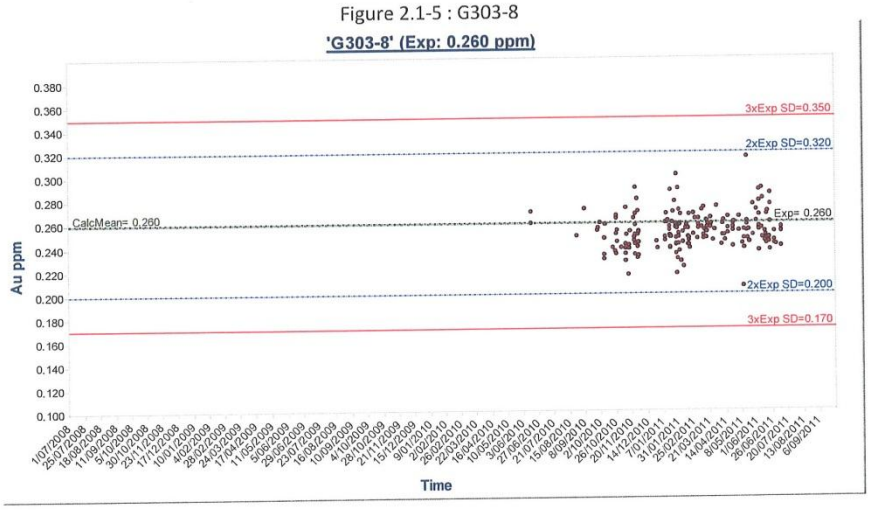

Table 2.1-3 : Outliers excluded from chart

StandardID	LabCode	Batch_No	Batch_Date	SampleID	Method	Expected	Au ppm	% Diff
G303-8	McPHAR	11-1050	26/05/11	EX621611	FA_AAS	0.26	1.63	-527

Figure 2.1-6 : G307-4

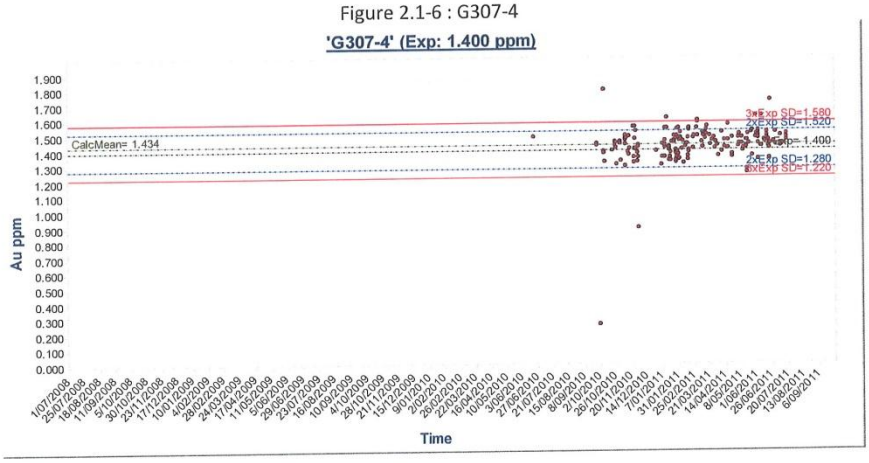

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Figure 2.1-7 : G399-2

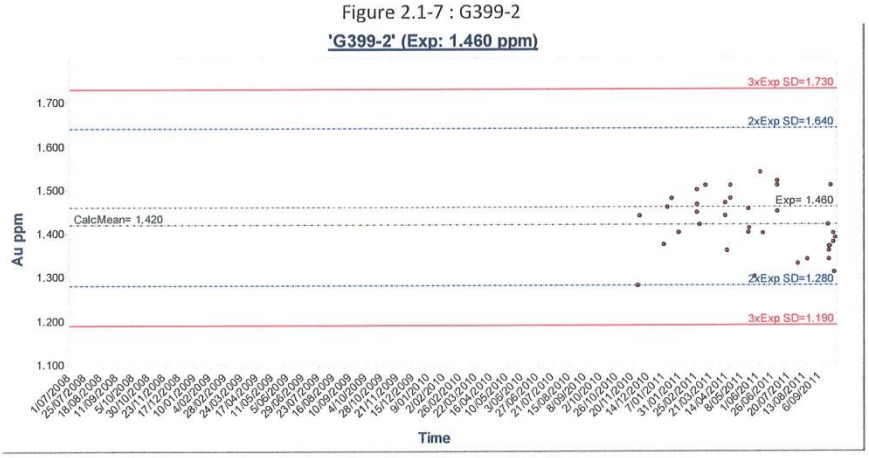

Figure 2.1-8 : G901-5

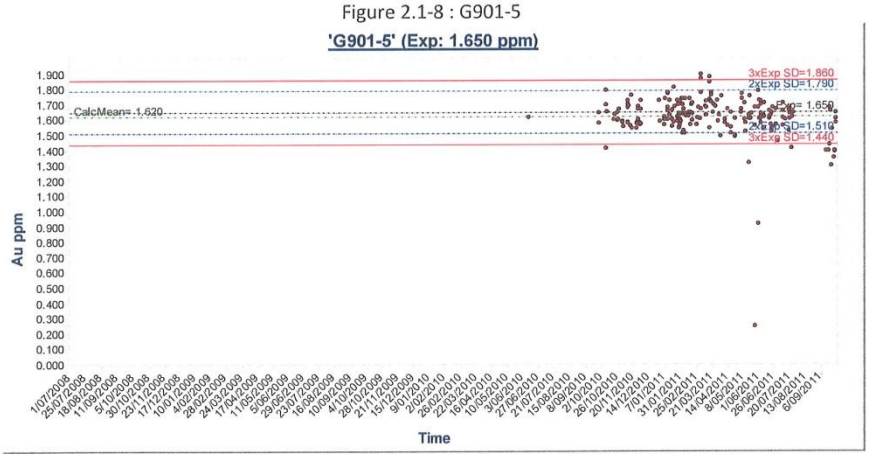

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Figure 2.1-9 : G907-2

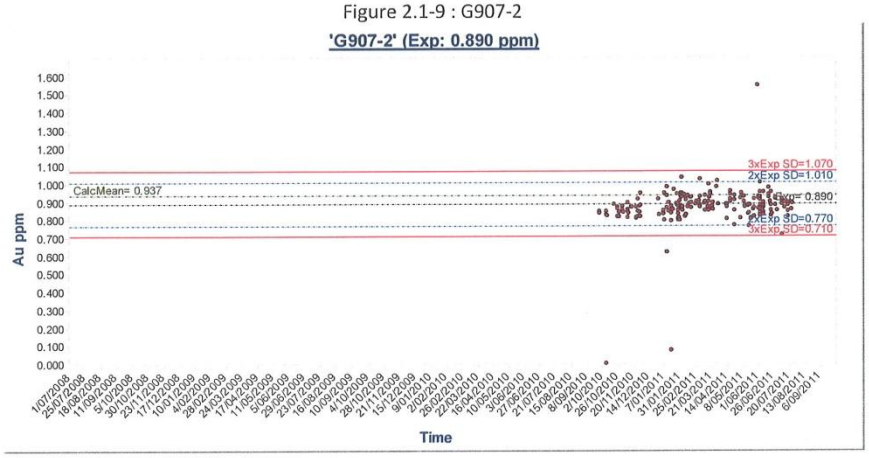

Table 2.1-4 : Outliers excluded from chart

StandardID	LabCode	Batch_No	Batch_Date	SampleID	Method	Expected	Au ppm	% Diff
G907-2	SGS_MAS	PH-MAS-FRC-11-005	23/09/11	EX637264	FAA505	0.89	6.3	-608
G907-2	McPHAR	11-0550	22/03/11	EX641571	FA_AAS	0.89	7.06	-693

Figure 2.1-10 : G907-3

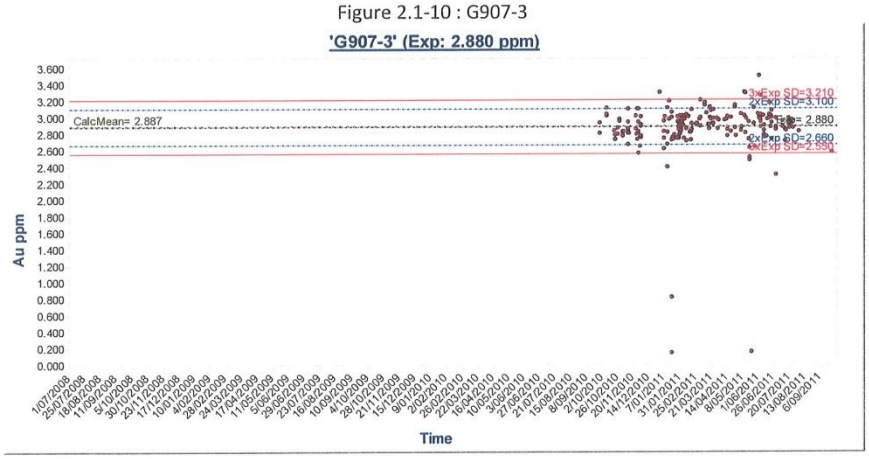

Table 2.1-5 : Outliers excluded from chart

StandardID	LabCode	Batch_No	Batch_Date	SampleID	Method	Expected	Au ppm	% Diff
G907-3	McPHAR	11-1440	09/07/11	EX636421	FA_AAS	2.88	6.63	-130

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Figure 2.1-11 : G907-8

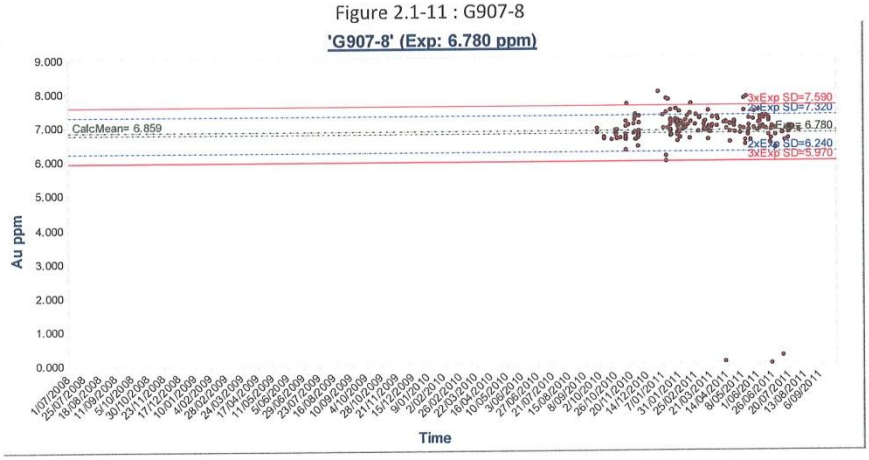

Figure 2.1-12 : Rocklabs Standards

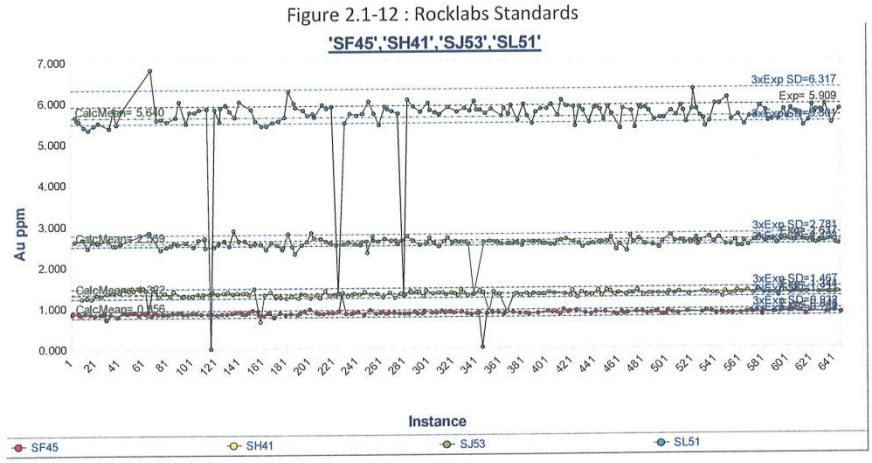

Table 2.1-6 : Outliers excluded from charts

StandardID	LabCode	Batch_No	Batch_Date	SampleID	Method	Expected	Au ppm	% Diff
SF45	SGS_MAS	PH-MAS-FRC-11-077	30/08/11	EX658709	FAA505	0.848	0.03	96
SF45	SGS_MAS	PH-MAS-FRC-11-049	16/08/11	EX645042	FAA505	0.848	1.32	-56
SH41	SGS_MAS	PH-MAS-FRC-11-025	04/08/11	EX652694	FAA505	1.344	0.66	51
SH41	McPHAR	11-1546	22/07/11	EX627919	FA_AAS	1.344	0.836	38
SH41	SGS_MAS	PH-MAS-FRC-11-084	31/08/11	EX653975	FAA505	1.344	0.9	33
SH41	SGS_MAS	PH-MAS-FRC-11-079	01/09/11	EX658475	FAA505	1.344	0.9	33
SJ53	SGS_MAS	PH-MAS-FRC-11-073	30/08/11	EX663701	FAA505	2.637	1.32	50
SL51	SGS_MAS	PH-MAS-FRC-11-023	01/08/11	EX652481	FAA505	5.909	0.01	100
SL51	SGS_MAS	PHMAS-FRC-11-0379	23/08/11	EX663147	FAA505	5.909	1.3	78
SL51	SGS_MAS	PH-MAS-FRC-11-047	16/08/11	EX653402	FAA505	5.909	1.34	77

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Figure 2.1-13 : SF45

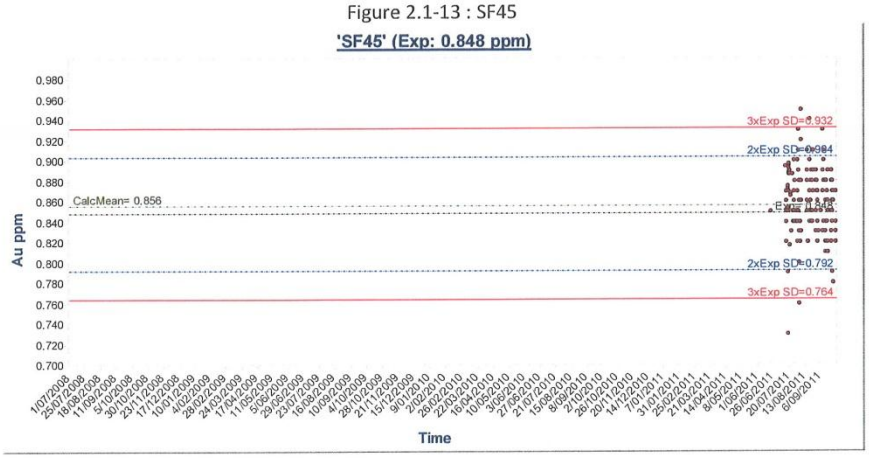

Figure 2.1-14 : SH41

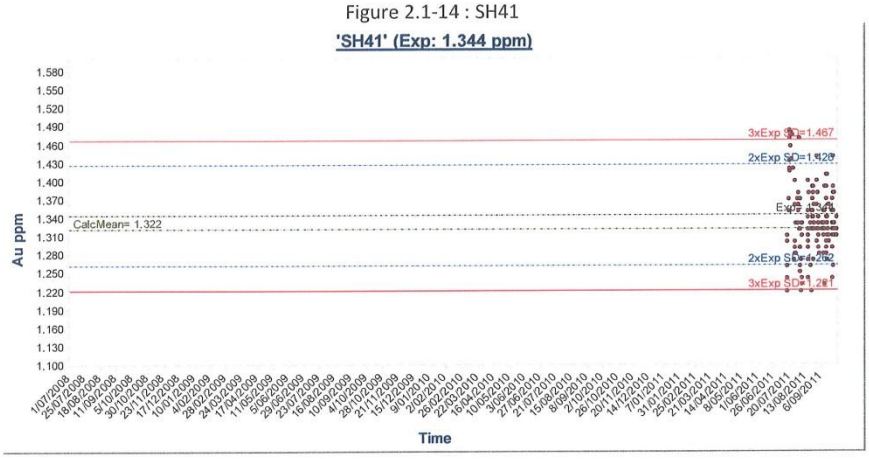

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Figure 2.1-15 : SJ53

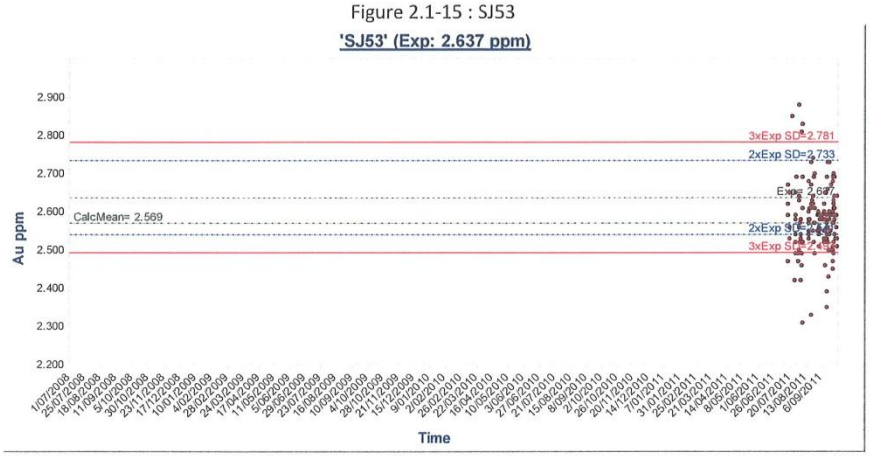
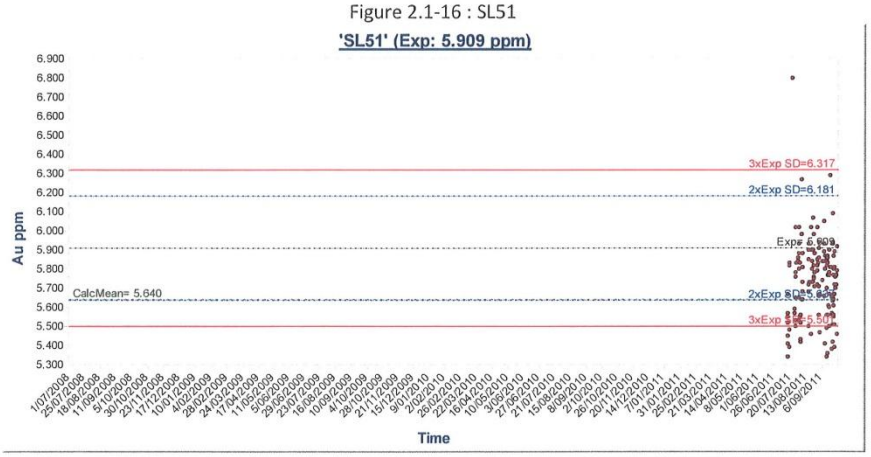


Figure 2.1-16 : SL51



ROCKSOLID DATA CONSULTANCY

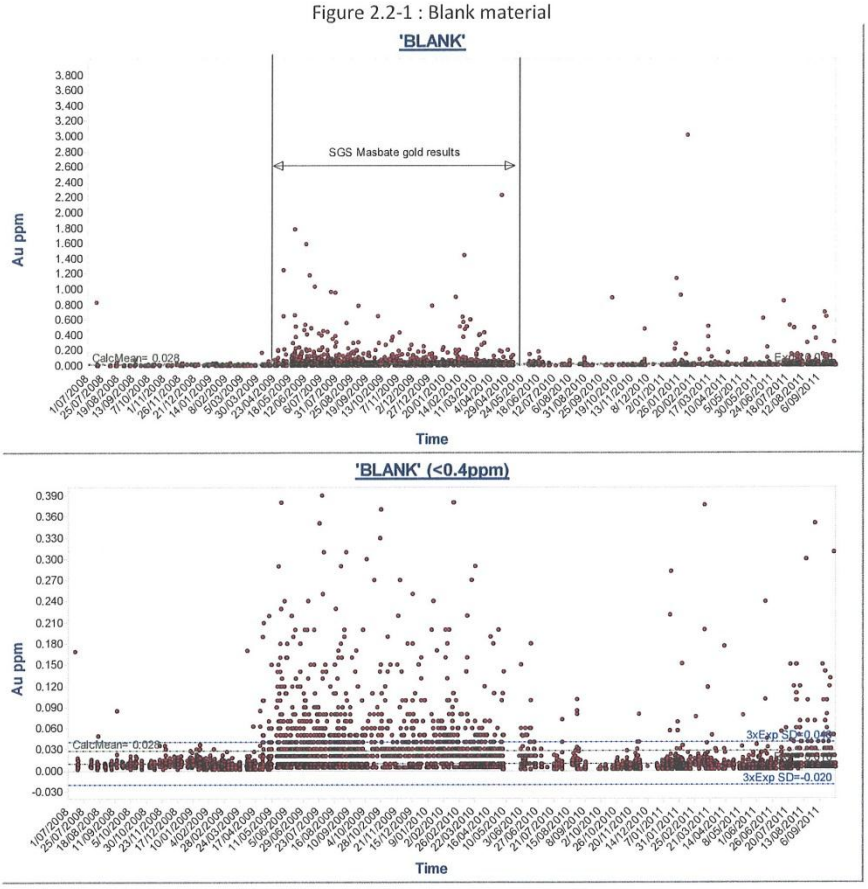
2.2 Blanks

Filminera used blank material during resource and grade control drilling campaigns to test for contamination during laboratory preparation and analysis. Blanks were inserted with drill samples at a rate of 1:40 samples. Blanks analysed by SGS between May 2009 and May 2010 during the RCGC drilling campaign show elevated gold levels, potentially contamination, however this might also be due to the type of material submitted as a"blank"; further investigation is required. Overall 27% of SGS blank results were >0.03ppm Au. The McPhar results perform very well with only 3% of results >0.03ppm Au, some of which may be attributable to standard mis-identification error. Outliers above 0.03 ppm Au are listed in the accompanying spreadsheet FRC_QC Report_20111219.xlsx

Table 2.2-1 : Resolute Blanks Summary

Standard	LabCode	No. of Samps	Expected Au ppm	Mean Au ppm	Min Au ppm	Max Au ppm	SD	CV	% Samps <=0.03
BLANK	McPHAR	2332	0.01	0.014	0.0025	3.015	0.08	5.828	97%
BLANK	SGS_mas	2304	0.01	0.043	0.005	2.23	0.115	2.669	73%
BLANK	Total	4636	0.01	0.028	0.0025	3.015	0.1	3.534	85%

Figure 2.2-1 : Blank material



ROCKSOLID DATA CONSULTANCY

3 Repeats

3.1 Field Duplicate

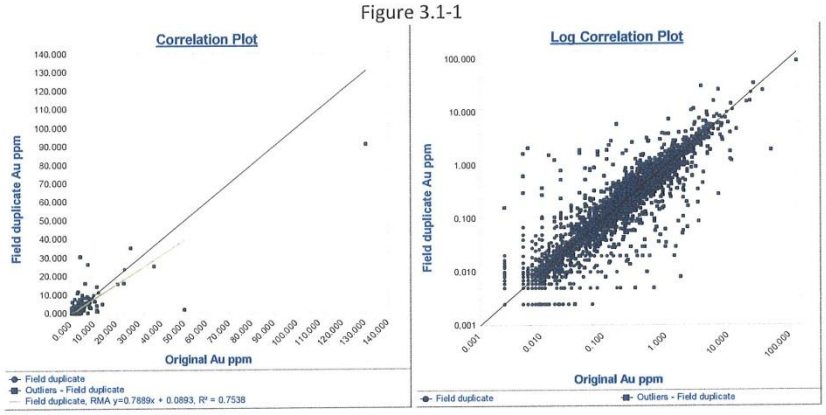
A total of 4658 field duplicate gold results from McPhar and SGS are included in the dataset. Samples sent to McPhar and SGS were analysed by the same method and in the same batch as the original sample.

The duplicate assay results exhibit generally good correlation with the original assays. A bias towards the original analysis is present in >0.1ppm range and a bias towards the repeat analysis is present in the <0.1ppm range. A number of samples returned gold assays indicative of potential sampling or laboratory preparation, analysis or reporting error, or may be a reflection of coarse gold present at the deposit. Outliers are listed in the accompanying spreadsheet FRC_QC Report_20111219.xlsx

Table 3.1-1

Range Au	No of Samples	mean Au1	mean Au2	SD Au1	SD Au2	CV Au1	CV Au2	mean HRD
0-0.1	2061	0.03	0.05	0.03	0.12	0.85	2.69	-3.56
0.1-0.3	1078	0.18	0.21	0.06	0.26	0.31	1.23	0.36
0.3-1	1056	0.54	0.56	0.19	0.35	0.34	0.63	2.72
1-10	448	2.24	2.25	1.48	2.59	0.66	1.15	7.43
10-1000	15	26.62	16.98	30.87	22.84	1.16	1.35	31.61
TOTAL	4658	0.48	0.47	2.39	1.88	4.98	4.03	-0.06

Figure 3.1-1



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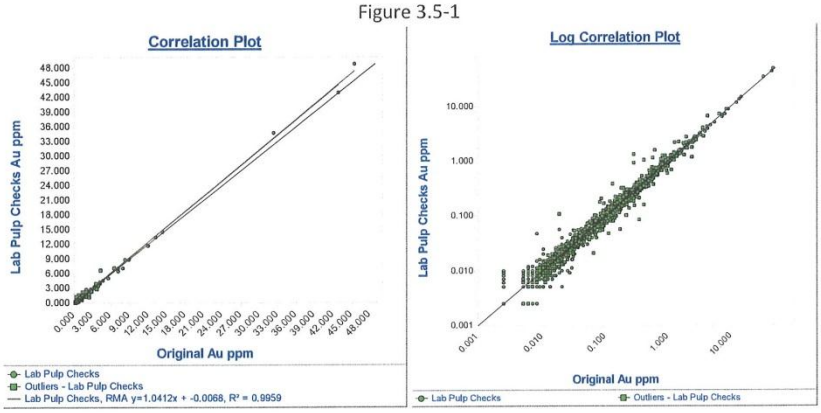
3.2 Laboratory Duplicate

A total of 2633 analytical repeats were reported by the laboratories for drill samples. The statistics and charts demonstrate good repeatability of the McPhar and SGS fire assay procedure. It is evident mean HRD of 0.1% that the variability does not have a consistent bias.

Table 3.5-1

Range Au	No of Samples	mean Au1	mean Au2	SD Au1	SD Au2	CV Au1	CV Au2	mean HRD
0-0.1	1716	0.03	0.03	0.02	0.03	0.97	1.01	0.04
0.1-0.3	444	0.18	0.18	0.06	0.09	0.31	0.47	0.22
0.3-1	348	0.55	0.55	0.19	0.22	0.36	0.41	0.23
1-10	119	2.14	2.15	1.61	1.65	0.75	0.77	0.12
10-1000	6	26.62	27.64	15.47	16.51	0.58	0.6	-1.3
TOTAL	2633	0.28	0.28	1.54	1.6	5.57	5.71	0.1

Figure 3.5-1


Correlation Plot
Lab Pulp Checks Au ppm
Original Au ppm
Lab Pulp Checks
Outliers - Lab Pulp Checks
Lab Pulp Checks, RMA y=1.0412x + -0.0068, R² = 0.9959
Log Correlation Plot
Lab Pulp Checks Au ppm
Original Au ppm
Lab Pulp Checks
Outliers - Lab Pulp Checks

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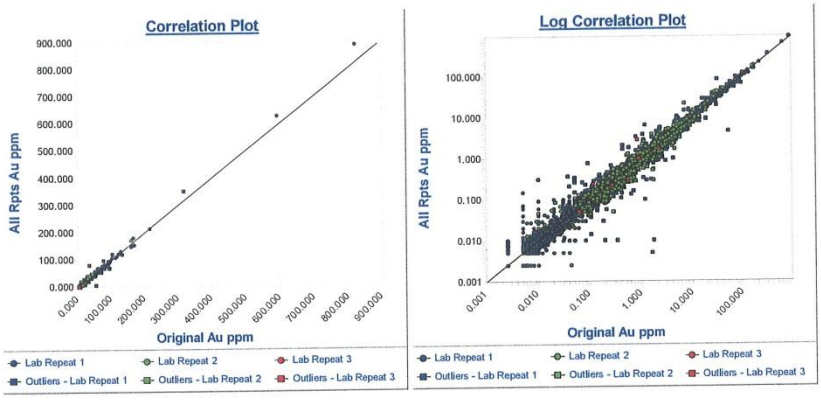
3.3 Laboratory Repeat

Another type of duplicate present in the resource dataset are laboratory analytical repeats. A total of 10506 analytical repeats were reported by McPhar and 10367 reported by SGS. The statistics and charts demonstrate the low variability of the results. A few outliers exist which might be caused by Filminera data capture errors or laboratory analysis or reporting errors. Outliers are listed in the accompanying spreadsheet FRC_QC Report_20111219.xlsx

Table 3.5-1

Range Au	No of Samples	mean Au1	mean Au2	SD Au1	SD Au2	CV Au1	CV Au2	mean HRD
0-0.1	6907	0.03	0.03	0.03	0.03	0.83	0.92	-0.56
0.1-0.3	3766	0.18	0.18	0.06	0.07	0.31	0.36	0.47
0.3-1	4480	0.57	0.57	0.19	0.21	0.33	0.37	0.37
1-10	5168	2.76	2.78	1.96	2.05	0.71	0.74	0.27
10-1000	552	27.89	28.35	49.67	52.88	1.78	1.86	-0.13
TOTAL	20873	1.59	1.6	9.29	9.78	5.85	6.1	0.04

Figure 3.5-1



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4 Particle Size Analysis

Screen testing is an analysis of the particle size of a prepared sample, which is a good indicator of the performance of the laboratory's sample preparation techniques. The particle size results are returned as a percentage of the sample which will pass through a specific screen size, in this case 75 microns. The results reported by McPhar Laboratory and SGS Laboratory since 2008 indicate acceptable pulverisation.

Table 4-1 : McPhar Laboratories Particle Size Analysis

Month	No. of samples	Overall Month Ave.	>=85% passing 75um	>=80 and <85 % passing 75um	<80% passing 75um
Jul-08	80	93.62%	100%	0%	0%
Aug-08	93	95.18%	100%	0%	0%
Sep-08	135	94.69%	100%	0%	0%
Oct-08	151	94.63%	100%	0%	0%
Nov-08	213	95.57%	100%	0%	0%
Dec-08	171	95.65%	100%	0%	0%
Jan-09	406	96.20%	100%	0%	0%
Feb-09	231	96.65%	100%	0%	0%
Mar-09	172	95.23%	100%	0%	0%
Apr-09	338	95.19%	100%	0%	0%
May-09	78	93.68%	100%	0%	0%
May-10	34	93.80%	100%	0%	0%
Jun-10	200	89.66%	96%	0%	5%
Jul-10	65	92.27%	100%	0%	0%
Aug-10	126	93.96%	100%	0%	0%
Sep-10	59	93.42%	100%	0%	0%
Oct-10	124	92.96%	100%	0%	0%
Nov-10	266	93.73%	100%	0%	0%
Dec-10	19	97.95%	100%	0%	0%
Jan-11	370	97.23%	100%	0%	0%
Feb-11	281	97.49%	100%	0%	0%
Mar-11	315	98.15%	100%	0%	0%
Apr-11	297	97.58%	100%	0%	0%
May-11	370	96.98%	100%	0%	0%
Jun-11	580	97.92%	100%	0%	0%
Jul-11	419	98.30%	100%	0%	0%
Aug-11	8	98.29%	100%	0%	0%
Total	5601	96.07%	100%	0%	0%

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Figure 4-1

McPhar Laboratories Screen Tests - 2008
Percentage of Samples meeting limits (p75um)
Average p75um
Month
Jan(0) Feb(0) Mar(0) Apr(0) May(0) Jun(0) Jul(80) Aug(93) Sep(135) Oct(151) Nov(213) Dec(171)
3.62 5.18 4.69 4.63 5.57 5.65
Average Pass
>=85%
>=80 and <85%
<80%
McPhar Laboratories Screen Tests - 2009
Jan(406) Feb(231) Mar(172) Apr(338) May(78) Jun(0) Jul(0) Aug(0) Sep(0) Oct(0) Nov(0) Dec(0)
6.20 6.65 5.23 5.19 3.68
McPhar Laboratories Screen Tests - 2010
Jan(0) Feb(0) Mar(0) Apr(0) May(34) Jun(200) Jul(65) Aug(126) Sep(59) Oct(124) Nov(266) Dec(19)
3.80 9.66 2.27 3.96 3.42 2.96 3.73 7.95
McPhar Laboratories Screen Tests - 2011
Jan(370) Feb(281) Mar(315) Apr(297) May(370) Jun(580) Jul(419) Aug(8) Sep(0) Oct(0) Nov(0) Dec(0)
7.23 7.49 8.15 7.58 6.98 7.92 8.30 8.29

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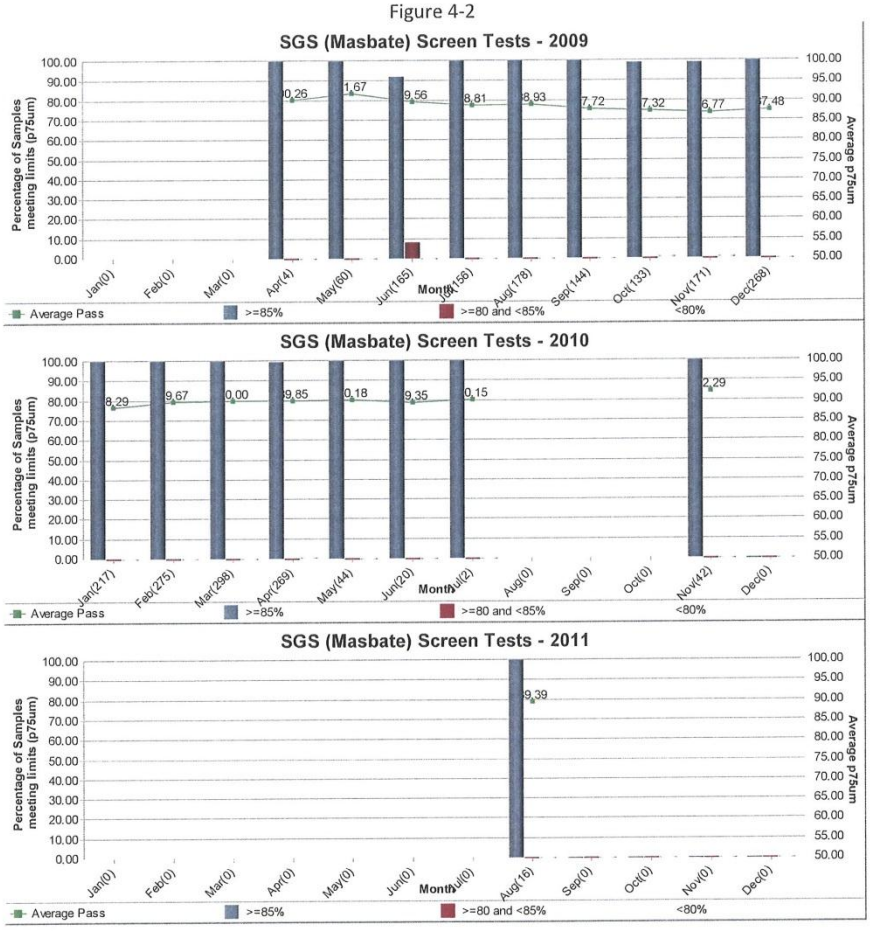
Table 4-2 : SGS Masbate Laboratory Particle Size Analysis

Month	No. of samples	Overall Month Ave.	>=85% passing 75um	>=80 and <85 % passing 75um	<80% passing 75um
Apr-09	4	90.26%	100%	0%	0%
May-09	60	91.67%	100%	0%	0%
Jun-09	165	89.56%	92%	8%	0%
Jul-09	156	88.81%	99%	1%	0%
Aug-09	178	88.93%	99%	1%	0%
Sep-09	144	87.72%	100%	0%	0%
Oct-09	133	87.32%	99%	0%	1%
Nov-09	171	86.77%	99%	0%	1%
Dec-09	268	87.48%	100%	0%	0%
Jan-10	61	87.42%	100%	0%	0%
TOTAL	1340	88.19%	99%	1%	0%

Figure 4-2



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30.2 Appendix 2

30.2.1 Risk Matrix

Likelihood		Impact
Number	Descriptor	
1	Almost certain	Occurs more than once per month
2	Likely	Occurs once every month - 1 year
3	Possible	Occurs once every 1 year to 10 years
4	Unlikely	Occurs once every 10 years to 100 years
5	Rare	Occurs less than once every 100 years

Severity		Impact
Number	Descriptor	
1	Insignificant	> 50% Reduction in production for < 1 week
2	Minor	> 50% reduction in production for < 1 month
3	Moderate	> 50% reduction in production for < 6 months
4	Major	Total loss of production for < 1 month OR > 50% reduction in production for < 12 months
5	Catastrophic	Total loss of production for > 6 months

Likelihood of the Consequence		Maximum Reasonable Consequence				
		Insignificant	Minor	Moderate	Major	Catastrophic
		1	2	3	4	5
Almost Certain	1	Low	Medium	High	Extreme	Extreme
Likely	2	Low	Medium	High	Extreme	Extreme
Possible	3	Low	Medium	High	High	Extreme
Unlikely	4	Low	Medium	Medium	High	Extreme
Rare	5	Low	Low	Medium	Medium	High